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PROPOSED METHOD FOR REDUCING MINERAL WASTE IN THE WISCONSIN ZINC DISTRICT, WISCONSIN

BY

WILL. H. COGHILL and C. O. ANDERSON

Published in cooperation with the
Missouri School of Mines and Metallurgy and the
Wisconsin Mining School

WASHINGTON
GOVERNMENT PRINTING OFFICE
1922

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First edition, November, 1922.

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PROPOSED METHOD FOR REDUCING MINERAL WASTE IN THE WISCONSIN ZINC DISTRICT, WISCONSIN.

By Will H. Coghill and C. O. Anderson.

INTRODUCTION.

The work described in this report, prepared in cooperation with the Missouri State school of mining and metallurgy and the Wisconsin State mining school, was part of an investigation by the Bureau of Mines in regard to metal losses in the mining and treatment of the lead and zinc ores of the Mississippi Valley. The text of the report is divided under three headings, as follows: (1) Scope and importance of investigation; (2) tabling of fine tailings; and (3) flotation of low-grade table concentrates and of jig concentrates.

The Wisconsin zinc and lead district comprises an area about 60 miles long, north and south, and about 30 miles wide, east and west. It lies in the southwest corner of Wisconsin, and extends a short distance into northern Illinois and eastern Iowa. Formerly the district was known best as a producer of lead, but is now an important source of zinc, the present output of lead being only a very small fraction of the output of zinc.

The climatic conditions and geographic location of the district are nearly ideal for mining operations. Most of the mines and mills are on flat, rolling farm land, and the various groups of mines are near railroads, so that transportation of ore is not difficult. The roads are generally good or are being improved through the use of mill tailings.

This report does not attempt to describe, except in a general way, milling conditions and milling procedure;[1] it discusses work done by the authors in an effort to reduce the mineral waste. To be sure, this waste is primarily zinc, but attention is called also to possibilities of saving and utilizing the lead, the marcasite, and the limestone tailing.

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[1] For an account of conditions as they were in 1914, see Wright, C. A., Mining and milling of lead and zinc ores in the Wisconsin district, Wis.: Tech. Paper 95, Bureau of Mines, 1915. 39 pp.

Figures are given showing the extent of zinc losses, but they are presented for purposes of information rather than to justify this report. Also some figures are presented to show the possible benefit from proposed methods of treatment.

For a long time operators have known that they were losing much zinc in the fine ore. Hence, the problem put before the writers was not necessarily to ascertain the annual loss of metal, but rather to devise a method of recovering the fine zinc; that is, to separate it from the marcasite, "oil rock," and other gangue constituents.

Ever since milling to recover zinc began the marcasite associated with the sphalerite has given the Wisconsin district an undesirable reputation. Separation of the two minerals has been difficult. A fairly satisfactory process of magnetic separation for grading up jig concentrates contaminated with marcasite has been in operation for some time, but the fines are not amenable to this process and have been wasted. The feasibility of installing the writers' process for the commercial recovery of fine zinc must be determined by the specific conditions at each plant.

The investigation was not limited to the losses in the sludge and fine tailings, but included the treatment of jig concentrates by flotation in order to get a higher recovery of zinc than is obtained at present by roasting and magnetic separation.

No mill tests were made nor was the work described in this report tried on a commercial scale, but the results obtained in the laboratory were gratifying, gave promise of being duplicated in the mill, and seemed to justify the erection of a small commercial plant. Of course, more laboratory work could be done advantageously after some of the difficulties of a commercial plant have been definitely ascertained.

COOPERATIVE ARRANGEMENT.

In November, 1920, at a meeting of the mine operators of the southwestern Wisconsin zinc district with United States Bureau of Mines officials at Galena, Ill., it was decided to undertake an investigation at Platteville, Wis., in an effort to save the zinc being wasted in the fine tailings. The work was begun early in January, 1921, under the following arrangement. The Wisconsin mining school, of Platteville, furnished the office and laboratory space, also light, power, gas, heat, and some of the equipment necessary for the work. Most of the time the mine operators took turns in furnishing a man, one of their technical staff, to assist in the work. In addition, the mining companies paid the assaying charges, cooperated cordially with the bureau men in giving information, bringing samples to the laboratory, and taking the bureau men to and through a number of the mills. The local assayer, Mr. C. W. Snow, did the analytical work, a total of 1,325 assays, at 25 per cent below the customary price. Mr. F. L. Walters, who made the chemical determinations, cooperated heartily in getting returns in the shortest possible time. The electric light company of Platteville supplied some of the motors for the machines. The Bureau of Mines supplied the services of two of its men, as well as some laboratory equipment.

The work covered a period of about eight months. One of the writers spent a month at the Forest Products Laboratory, Madison, Wis., in a study of oils best suited to floating Wisconsin sphalerite in the presence of marcasite.

ACKNOWLEDGMENTS.

The writers greatly appreciate the cordial cooperation of all the parties mentioned above. Mr. H. B. Morrow, director of the mining school, and his faculty rendered every assistance possible. In addition thanks are tendered to Mr. C. R. Kuester, of the Vinegar Hill Zinc Co., for valuable assistance rendered for several months in the laboratory. The assistance given by many others, who spent brief periods in the laboratory, is highly valued. Especial mention is made of the work done by Mr. J. H. Carney, of the W. A. Butchart Co., of Denver, Colo., who came from the Tri-State Zinc District and spent two weeks in determining factors involved in the tabling of the sludge and fine tailing. Thanks are also due Dr. L. F. Hawley and Mr. H. N. Calderwood, jr., of the Forest Products Laboratory, Madison, Wis., for the direction and advice given in the work on oils.

Considerable interest was manifested by the mine operators in the work throughout in spite of the fact that the slump in the zinc market had closed almost all of the mines and mills. The chief difficulty encountered in the work was the procuring of fresh samples because of the mills being idle.

SCOPE AND IMPORTANCE OF INVESTIGATION.

THE ORE.

Primarily, the ore investigated consists of sphalerite and galena in a marcasite-limestone gangue. Crushing to "mill size," that is, to $\frac{7}{16}$ inch, generally liberates the sphalerite and galena to a satisfactory degree and leaves some of the marcasite locked with the lime. At some plants these locked grains caused trouble in milling because their specific gravity too closely approaches that of sphalerite. The problem of chatty (locked) zinc or lead mineral is negligible in this district. Barite is an occasionally gangue constituent and some quartz is usually present, but the gangue constituent most deleterious to flotation is the so-called "oil rock," a shaly, flinty-looking material with a specific gravity of about 2.

Bain[2] has described this oil rock as "shaly partings which are hightly fossiliferous and in part at least carbonaceous." Some fragments can be lit with a match and give off a petroleum odor on burning. The readiness with which the rock floats is evidenced by the fact that small amounts of oil have been distilled from it. Oxidized zinc, or "dry bone," is found in some mines, but it can not be recovered by the method proposed in this report.

IMPORTANCE OF THE DISTRICT.

The following quotation[3] gives some idea of the importance of the district: "A total of 68 mills and 9 separating plants were in operation during 1917, as against 82 mills and 14 separating plants in 1916."

In 1917[4] the district contributed 9 per cent of the production of zinc in the United States, whereas Kansas contributed 2.8 per cent, Oklahoma 12 per cent, and Missouri 18.6 per cent.[4]

IMPORTANCE OF IMPROVEMENTS IN METHODS OF MILLING.

The Bureau of Mines has described mining and milling practice in the Wisconsin district[5] and in the Joplin district.[6] Robert W. Johnson Associates, engineers, Pitcher, Okla., has issued a valuable handbook on milling practice in the Oklahoma-Kansas district. Those who desire detailed descriptions of ores and methods of treatment not contained in this paper are referred to the reports mentioned.

Because of the lack of correlation in dates and authorship, it is difficult to make accurate deductions from these reports, but the mills in the Wisconsin district, in spite of the zinc content of the mill feed being lower, surpass those of the Missouri-Kansas-Oklahoma (Tri-State) district in the percentage of zinc recovered. Though the zinc recovery is somewhat higher, the iron content is so great that the concentrates are of much lower grade and almost without exception have to be treated again by roasting and magnetic separation.

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[2] Bain, H. F., Zinc and lead deposits of the upper Mississippi Valley: U. S. Geol. Survey Bull. 294, 1906, p. 25.

[3] Kennedy, J. E., Zinc mining in Wisconsin: Eng. and Min. Jour., vol. 105, Jan. 12, 1918, p. 69.

[4] Siebenthal, C. E., Zinc in 1917: U. S. Geol. Survey, Mineral Resources of the United States, 1917, part 1, p. 864.

[5] Wright, C. A., Mining and milling of lead and zinc ores in the Wisconsin district, Wisconsin: Tech. Paper 94, Bureau of Mines, 1915, 39 pp.

[6] Wright, C. A., Mining and treatment of lead and zinc ores in the Joplin district, Missouri: Tech. Paper 11, Bureau of Mines, 1913, 15 pp.

The jig is the only concentrator found in the Wisconsin mills, whereas in the Tri-State field jigs are often supplemented by concentrating tables and sometimes by flotation machines. However, these supplementary machines do not offset the ill effect of the harder gangue and the greater number of locked grains (chatty mineral). The Wisconsin ore is very easy to crush, and, in consequence, grinding costs are only a small fraction of those in the Tri-State district.

The Wisconsin operators are not satisfied with their milling practice and wish to increase the recovery of zinc. The discussion relative to Table 1 shows what degree of improvement should be made.

TABLE 1.—*Data on milling in Wisconsin in 1917.*[a]

Crude ore (mill feed)	tons	[b]2,711,250
Sulphides (jig concentrates)	do	223,916
Zinc content	do	73,444
Assay of mill feed, assuming 70 per cent recovery	percentage of zinc	3.87
Tailings	tons	2,487,334

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[a] Dunlop, J. P., and Butler, B. S., Silver, copper, lead, and zinc in the Central States in 1918: U. S. Geol. Survey, Mineral Resources, 1918, pp. 120 to 180.

[b] Official figures reduced 10 per cent to deduct for bowlders.

If it be assumed that the recovery shown by "zinc content" in Table 1 was 70 per cent, then the zinc left in the tailings amounted to 31,476 tons. The tailings would assay 1.26 per cent Zn. If, on passing the tailings over a 20-mesh screen, 65 per cent of this zinc should pass through with 25 per cent of the charge, these fines would amount to 621,833 tons and contain 20,459 tons of zinc. The fines would assay 3.29 per cent Zn.

From a current product such as that represented by this average, 80 per cent of the zinc in the fines could be recovered by the method proposed by the writers. It would amount to 16,367 tons of zinc, an increase in recovery of $(0.80\times0.65\times0.30=0.156)$ 15.6 per cent, thus bringing the assumed recovery of 70 per cent up to 85.6 per cent. On a $40 zinc ore market this added recovery of zinc would amount to more than one million dollars annually.

In a second assumption, let the mill recovery of zinc be 65 per cent instead of 70 per cent; then the zinc in the tailings—zinc content 1.59 per cent—would amount to 39,550 tons. The fines minus 20-mesh ore would amount to 621,833 tons and contain 25,707 tons of zinc. Then the fines would assay 4.13 per cent Zn. With a recovery of 80 per cent of the zinc in the fines, the zinc obtained would be 20,566 tons and the mill recovery would be brought to 83.2 per cent or an increase of 18.2 per cent. The value of the zinc recovered from the fines would be, on a $40 zinc ore market, more than one and one-third million dollars.

Statistical reviews convey the impression that the district is of importance only as a zinc producer, but the by-products deserve mention.

Besides the lead, the ore contains two constituents, marcasite and limestone, that can be segregated and marketed. Marcasite is the source of sulphuric acid, and ground limestone is required for agricultural purposes. The demand for tailings for roadbeds and for concrete already exists.

Until recently the sulphur fumes from the roasters at the magnetic separation plants went to waste, but now the district has two fully equipped acid plants. When the industry is fully developed the marcasite will become an asset rather than a liability. As the farmers come to appreciate the value of fine limestone for improving soils, any of it produced in milling will meet an increasing demand in agriculture.

ALLEGED LAXITY IN MILLING PRACTICE.

Wet concentration has been solely by jigs, and the failure to employ supplementary concentrating devices has brought criticism from mining men practicing elsewhere. The limitations of the jigs and the resulting losses of zinc in the fines are so evident that an uninformed observer at once declares that the milling is primitive and inadequate and that the fine ore should go to concentrating tables. These, however, have been tried repeatedly and have failed, not because they make poorer recoveries and lower-grade products than the jigs, but because the table concentrate is not amenable to further grading up by magnetic separation as applied to the jig concentrate. Whereas the latter is an ideal product for roasting and magnetic separation to remove the iron, the table concentrate is too fine and dusty. Thus table concentration has been a failure because of the absence of a grading-up system corresponding to the magnetic separation of jig concentrates. Static separators failed after a commercial trial. In short, the operators have not had a commercial process for separating marcasite from sphalerite in material finer than 40 mesh.

Having had free access to records, the writers know that for the last 10 years the operators have made determined efforts to improve recoveries, but these, stated in round numbers, are the discreditable conditions they face: After milling the recovery of zinc is less than three-fourths; after magnetic separation, two-thirds; and after smelting, one-half. Such results condemn the present metallurgy of zinc and indicate the urgent need for improvement in all phases of it.

Another unjust criticism is that the mill feed is rarely assayed. When mill tests are made, the concentrates, the tailings, and the sludge are sampled and these form the basis for calculating the grade of the feed. The feed is not sampled because attempts to sample it have given unsatisfactory results. In western mining camps where finer grinding is practiced—to, say, 100 mesh—sampling is a simple matter because 50 pounds per shift, taken properly, will be representative. When, however, the ore is ground to pass ½ or ⅜ inch mesh, as is customary in jigging, so large a feed sample is required that at small mills the cost of handling becomes excessive. At first thought the same argument may seem to apply to the sampling of tailings and concentrates, but it does not. The tailing is comparatively low grade, and the "nuggets" of ore have been removed by the jig, so that the taking of a fairly accurate sample is easy. The concentrates, being a small part of the whole, permit an error in sampling of, say, 1 per cent in zinc content, whereas the same error in the feed sample might vitiate the results by as much as 20 per cent. Moreover the grade of the concentrate is kept fairly uniform, whereas the grade of the feed may fluctuate so widely as to defy all sampling methods except the most precise.

CONSIDERATIONS DETERMINING THE PROBLEM.

The jig is a good concentrating device where its limitations are recognized and observed. Carrying farther the calculations for Table 1 shows that whereas the minus 20-mesh tailings assayed either 3.29 or 4.13 per cent Zn, the plus 20-mesh material would assay only 0.6 per cent Zn. Numerous assays made in the course of this work, on plus 20-mesh jig tailings, show 0.4 per cent to 0.7 per cent Zn. Thus in handling coarse material the jig did efficient work and made a good recovery, but in treating fine ore it failed; the fine grains not only did not yield their mineral content but made an undesirable load on the jig. The effectiveness of the jig in treating coarse zinc is shown by its recovery of approximately 90 per cent of the plus 20-mesh zinc. As galena concentrates are also made on the jig, the loss of fine lead in the jig tailings must be considered in any process intended to reduce the mineral wastes in the district.

As already noted, the chief drawback in the development of the Wisconsin zinc district has been the marcasite associated with the sphalerite; it is even more difficult to separate by gravity concentration than is the pyrite in the complex ores of the West. Three reasons for the trouble it gives are: (1) Its specific gravity is a little less than that of pyrite, (2) it breaks so that the mineral grains have a very uneven ("cockscomb") surface which retards their fall in water, and (3) some of it remains locked with the limestone and the composite grains have the same specific gravity as sphalerite. This intimate mixture of marcasite and limestone, called "sulphur rock," is the source of both lime and iron in the zinc concentrates.

From these facts one can easily see why the jig concentrates have an average grade of only about 35 per cent Zn. In some mills the grade of the jig concentrate falls as low as 20 per cent and much jig concentrate in the district assays less than 30 per cent Zn. Only by means of the roasting and magnetic separating plants are these mixtures graded up to make a zinc concentrate acceptable for smelting.

It is hoped that the marcasite will become an asset as a source of sulphuric acid, but modifications in milling will probably be necessary, because the separation of marcasite causes a loss of zinc in the roaster tailings that amounts to 8 per cent of the zinc in the charge.

The figures given above show that the mills are making a satisfactory recovery from the coarse feed but recovering very little, probably not more than one-fourth, of the sphalerite from the fines. Any substantial increase in recovery will necessitate treating the fines, commonly termed sludge, more efficiently. But this sludge is a fortuitous product that overflows crude settling boxes at the end of the jigs; the boxes are intended to remove only water and the solids that are fine enough to flow in a spout, and thus take part of the load off the tailing elevators. After the sludge has settled the water is returned to the mill. The sludge has about the same zinc content as the mill feed and represents about one-tenth of it; that is, one-tenth of the zinc is lost in the sludge, which, naturally, has received most of the attention in former efforts to prevent losses.

Flotation has been tried with great persistence, but the results have been erratic and unsatisfactory. The sludge from any two mills may vary widely in a sizing test, as the settling boxes are of no standard size, have no definite regulation, and are at best a crude device. Some mills send far more fines into the tailing elevator than do others. Some sludges may show as much as 60 per cent of their bulk on 100 mesh, whereas others may show practically nothing on this mesh.

The writers gave little attention to the flotation of sludge, because they believed that flotation would not prove effective because, first the sludge contains some zinc grains that are too coarse, and, second, it is badly contaminated with clay, "oil rock," and foul water. If means were devised for treating it, there would still be heavy losses in the fines sent to the tailing plates. When all considerations are taken into account it follows, therefore, that an adequate process must treat all material finer than 20 mesh—both the fines going to the tailing elevators and the solids in the overflow water.

A ruling factor in the selection of such a process is that the mines are short lived, averaging only about two years; consequently the mills can not be costly. The mills are small, simply built, and contain no such elaborate equipment as is found in western mills. All factors being considered the scheme that seems most feasible is to separate the fines that pass a 20-mesh screen from all the material that now flows from the tailing and sludge spouts of the mills, to treat these fines on concentrating tables, and to make not a marketable smelting product but a low-grade concentrate to be shipped to a central flotation plant where adequate grinding and flotation machinery are available for final treatment. This, briefly, is the fundamental idea the authors kept in mind in their work.

The tabling required at each mill would be what is commonly termed roughing; that is, the tables would require little skill and attention because they would make only low-grade tailing and low-grade concentrate. Such a concentrate would be granular, easy to transport and readily amenable to flotation after grinding; it would be of about the same grade (from 20 per cent zinc to about 30 per cent zinc in some mills) as much of the jig concentrate now shipped to the roaster.

The tabling of this material calls for a few comments. A table in operation develops three marked fans: The upper, of marcasite; the middle, of sphalerite; and the lower, of limestone. At first sight these fans seem to contain clean products. Indeed, a large part of the iron can be cut out with moderate loss of zinc, and a similar cut can be made into the limestone, but the sphalerite fan is not as clean as it seems. It is permeated with iron from one side and with lime from the other, so that its grade is too poor for smelting. To get a high-grade clean product of a mineral having a specific gravity that makes it lie between two other minerals on a concentrating table is difficult. The writers believe that such separation is physically possible, but that the equipment required would be too elaborate. Tables, however, will remove much of the iron from the zinc, thus giving a zinc concentrate that can profitably be hauled to a central flotation plant. The zinc content of this concentrate can be maintained at about a certain figure, regardless of the iron content of the table feed.

Another advantage which can not be too strongly emphasized of such tabling is that some galena will be recovered in the iron fan. At some plants the revenue from the lead would pay the cost of tabling. Finally, tabling would remove oil rock, clay, and foul water that would interfere with flotation.

One important fact to be kept in mind is that the fines in the scheme proposed are the result of the primary crushing in the mill and not a product of secondary crushing. The jig feed is ground through $\frac{7}{16}$ inch at the head of the mill and there is no further grinding; in consequence, there is no large amount of excessively fine material, and high table recoveries are possible. If this ore were reground, as in the Tri-State field, the proportion of fines in the form of slime would be far greater and the table recoveries would be low. Furthermore, the Wisconsin ore is softer than that in the Tri-State district, and the zinc does not become as fine in crushing to a given size. The jig has effectively removed the zinc in the coarse material, but has failed to get a considerable part of that in the fines, which originally are somewhat richer than the coarse material, as the zinc crushes a little more easily than the limestone gangue and segregates in the fines.

The laboratory work and the results obtained are presented under two general headings: (1) Tabling of fine tailings, and (2) flotation of low-grade table concentrates and of jig concentrates.

TABLING OF FINE TAILINGS.

SCREEN ANALYSES OF TAILING PILES.

After tabling had been determined as the first step in treating the fines, it was important to know at what mesh the tabling should begin. The decision would depend primarily upon the grade of the tailing in the various meshes.

As most of the mills were closed, the writers did not have the time to make, nor would they have been able to make, a systematic screen-analysis survey of the mill tailings and sludge; in consequence the following figures on screen analyses of tailings are drawn from various sources.

Wright[7] in 1913 presented some data that are contained in Table 2.

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[7] Wright, C. A., Mining and milling of lead and zinc ores in the Wisconsin district, Wis.: Tech. Paper 95, Bureau of Mines, 1915, pp. 30–32.

Table 2.—*Screen analyses of tailings, by Wright.*

Product.	Per cent of total weight.	Zinc.		Iron.	
		Assay, per cent.	Per cent of feed content.	Assay, per cent.	Per cent of feed content.
Composite No. 1	100.0	1.96	100.0	4.24	100.00
Through ⅜, on 20 mesh	56.9	.78	22.6	3.70	49.70
Through 20, on 40 mesh	8.1	2.44	10.1	4.00	7.65
Minus 40 mesh	35.0	3.77	67.3	5.20	42.65
Composite No. 2	100.0	1.72	100.0	4.60	100.00
Through ⅜, on 20 mesh	62.4	.79	28.4	3.77	51.10
Through 20, on 40 mesh	5.6	1.63	5.3	5.50	6.80
Minus 40 mesh	31.9	3.58	66.3	6.07	42.10
Composite No. 3	100.0	1.34	100.0	4.71	100.00
Through ⅜, on 20 mesh	75.0	.81	45.3	5.00	79.50
Through 20, on 40 mesh	4.2	1.07	3.3	3.70	3.25
Minus 40 mesh	20.8	3.31	51.4	3.90	17.25
Composite No. 4	100.0	1.18	100.0	3.60	100.00
Through ⅜, on 20 mesh	78.1	1.00	66.0	3.85	83.60
Through 20, on 40 mesh	2.1	.97	1.7	2.70	1.60
Minus 40 mesh	19.8	1.92	32.3	2.70	14.80
Composite No. 5	100.0	1.33	100.0	5.34	100.00
Through ⅜, on 20 mesh	62.6	.87	41.9	5.50	63.90
Through 20, on 40 mesh	9.6	1.70	12.2	4.90	9.10
Minus 40 mesh	27.8	2.20	45.9	5.30	27.00

The iron analyses are included simply to show that considerable iron sulphide (marcasite) is present. The plus 20-mesh material is very low in zinc and could not be retreated profitably by tabling because of the cost of the necessary crushing. Table 2 indicates that on the average 60 per cent of the zinc in the tailings is in minus 20-mesh material, which is about one-third of the total weight of the tailings. The results in this table indicate that tabling of the tailings should begin at a mesh not coarser than 20 (0.833 mm.).

In a test extending through October, 1915, the late Mr. H. A. Roessler, of the Vinegar Hill Zinc Co., obtained as an average the following figures for tailings from a number of the company's mills.

Table 3.—*Screen analyses of tailings, by Roessler.*

Product.	Per cent of total weight.	Zinc.	
		Assay, per cent.	Per cent o composite content.
Composite	100.0	1.29	100.00
On 1 mm. (17 mesh)	77.1	0.54	32.3
Through 1 mm., on 40 mesh	22.9 { 6.6	3.82 { 1.45	67.7 { 7.3
Through 40 mesh	16.3	4.77	60.4

These figures demonstrate the efficiency of the jig in recovering coarse mineral and show that it is not feasible to begin tabling with material coarser than 20 mesh; they also show that 67 per cent of the zinc is contained in the minus 1-mm. material, which by weight is 23 per cent of the total tailing.

Some screen analyses made by the writers are contained in Table 4.

TABLE 4.—*Screen analyses of tailings, by Coghill and Anderson.*

Product.	Per cent of minus 20 mesh.	Assay, per cent of zinc.	Per cent of total zinc in tailing.
Mill No. 1	28.50	3.70	75.4
Mill No. 2	20.75	3.45	66.1
Mill No. 3	14.20	2.50	49.4
Mill No. 4	17.90	3.30	60.0
Mill No. 5	33.00	2.80	65.0

In each test the plus 20-mesh material assayed between 0.4 per cent Zn and 0.75 per cent Zn. Consequently it was too low grade for table feed.

From these miscellaneous figures and from other assays and figures supplied by the mine operators, the writers decided that it would be most feasible to begin tabling the tailings at 20 mesh. It is important to note that no grinding is required to prepare the tailings for tabling.

FACTORS THAT INFLUENCE TABLING.

A review of many of the mill tests made by the mining companies and a consideration of the assays of a number of sludge samples with which the writers experimented showed that the material overflowing the settling boxes in the mills and going to the sludge ponds assayed from about 3 per cent to more than 6 per cent Zn; rarely, where the mines were unusually rich, the zinc assay was much higher. The sludge and tailings assay from about 3.5 per cent to 12 per cent Fe. It is believed that it can be safely stated in round numbers that the material available for tabling contains 65 per cent of the zinc now lost and this in a bulk of 25 per cent of all the tailing material now issuing from the mills. This feed to the table would assay 3 to 5 per cent zinc and one to more than two times as much in iron. This statement coincides approximately with that of some of the operators, who say: " We recover in our jig concentrates two-thirds of the zinc and one-third of the iron, and leave in the tailings and sludge one-third of the zinc and two-thirds of the iron." Many jig concentrates contain, by assay, twice as much zinc as iron, and the tailings carry twice as much iron as zinc.

The table feed proposed consists of the three sulphides, sphalerite, marcasite, and galena, the limestone gangue, and some quartz. In addition it contains the oil rock, all the clayey constituents of the original ore, and the foul water. When this material is fed to a concentrating table, the following products are found, beginning at the top of the concentrating end: (1) A thin streak of lead, (2) a broad band of iron blending into the band of zinc, (3) the zinc band, (4) the limestone gangue which contaminates part of the zinc band and shades off into a clean tailing, and (5) near the back-water side of the table, the oil-rock band.

The primary object of the tabling is to make a low-grade zinc concentrate for flotation. The grade depends on different factors, but it was decided to make at least a 20 per cent grade and higher if possible, as such a product could be shipped profitably to a central flotation plant; a great deal of the jig concentrates now profitably shipped to the roaster assay between 20 per cent and 30 per cent Zn. The table removes the foul water, the clayey material, and the oil rock, which would seriously interfere with flotation. Most investigators attempting to save the zinc in the fines of this district have failed because they aimed to send the sludge, without any preliminary treatment, direct to the flotation cell.

In spite of its low specific gravity (about 2), the oil rock can not always be removed completely from the zinc product because of its being locked with marcasite and limestone ("sulphur rock") and so forms grains of about the same specific gravity as sphalerite. These locked grains lower the grade of the table concentrate.

Another factor governing the grade is the extraction or recovery sought, or the leanness of the tailing. To get a tailing low in zinc some of the limestone must be allowed to go with the zinc product, thus lowering its grade.

Early in their investigation the authors thought that the slight difference in specific gravity of the zinc and iron sulphides would render impossible the production of a table concentrate of suitable grade from many sludges—that from any table feed carrying zinc and iron sulphides the concentrate would carry zinc and iron "graded up" in the same relations as in the feed. A table feed having a high iron content would yield a table product carrying far less than 20 per cent zinc. Three things, however, offset the proportional grading up of the iron: (1) The loss of fine iron as marcasite in the table tailing exceeds the loss of fine zinc; (2) some of the iron is contained in the gangue composition; and (3) an iron concentrate very low in zinc can be separated on the table at the same time as the zinc concentrate.

The lead may be separated in a concentrate by itself, or allowed to go into the iron concentrate which may be retabled to obtain the lead.

The granular zinc sulphide concentrate obtained, which is to serve as a flotation feed, is free from many objectionable constituents and except for the locked oil-rock is satisfactory for flotation.

TABLING ON LABORATORY-SIZE WILFLEY.

In the first tabling tests a small laboratory-size (12 by 30 inches) Wilfley table was used and the samples were dug from sludge ponds. Later tests were made on a quarter-size commercial table.

In obtaining samples for table feed it was difficult to decide in just what proportions to mix fine tailings and sludge, because of the lack of definite figures on the proportions of these products made in the mill. No two mills send the same equivalent amounts to the tailing elevator and to the sludge trough, and in order to make an accurate determination, figures for each individual mill would have to be obtained. The proportion of 2 parts of fine tailing to 1 part of sludge was adopted for a number of tests. If the relative amounts in the various meshes were changing it would not affect the tabling, as a classifier would be required in the mill.

In tests on the small Wilfley table, material must be repeatedly refed to the table in order to get results comparable to those on a larger. The deck is too small and the distance of travel of the grains too short to permit good classification, stratification, and separation, but with three treatments the results compare with those obtainable on a larger table.

Table 5, which records a number of small-table tests of samples taken from sludge ponds, presents many interesting facts. The zinc assays ranged from 4.4 per cent to nearly 8 per cent, and the iron assays varied even more widely. In some of these sludges the lead content, not given in the table, was appreciable and would be a source of revenue. The fineness of the sludge varied widely; in some tests there was practically nothing on 100 mesh and in others there was several per cent on 40 mesh.

No iron concentrate was made in any of these seven tests. Consequently in Nos. 1 and 7 the iron assay of the sulphide concentrate was more than 20 per cent Fe and the zinc assay was only 11.5 per cent and 14.6 per cent Zn. The necessity of removing the iron by table, if higher-grade zinc concentrates are desired, is evident.

The ratio of concentration varies from 2:1 to 5:1. Roughly speaking, the greater the ratio of concentration the higher the grade of concentrate, and the lower the ratio of concentration the higher the recovery and the lower the zinc content of the tailing. This is shown in Nos. 3, 4, and 5, where the grade of concentrate increases with increasing ratio of concentration and decreasing recovery.

The last column, "zinc and iron sulphide, per cent," shows the sulphide content of the "sulphide concentrate." This figure averaged close to 60 per cent (60 to 75 per cent) in all the table tests made. Evidently there was still about 25 to 40 per cent of gangue material in the product to be sent to flotation, but on the other hand the table feed carried about 80 per cent of gangue material and 20 per cent of sulphides. A low-grade zinc concentrate running 60 to 75 per cent sulphides and 25 to 40 per cent gangue is about the average to be expected from the tabling advocated in this report.

TABLE 5.—*Concentration tests, on small (12 by 30 inches) Wilfley table, of sludges to make low-grade zinc concentrate.*

Product.	Weight of product, grams.	Per cent of total weight.	Zinc.		Iron.		Lime (CaO).		Ratio of concentration.	Zinc extraction, per cent.	Zinc assay, per cent.	Zinc and iron sulphides, per cent.
			Assay, per cent.	Per cent of feed content.	Assay, per cent.	Per cent of feed content.	Assay, per cent.	Per cent of feed content.				
No. 1 sludge, 98.0 per cent minus 100 mesh:												
Table feed	6,710	100.00	4.40	100.00	8.80	100.00	21.00	100.00				
Sulphide concentrate	2,000	29.82	11.46	77.62	20.57	69.69			3.3:1	77.66	11.46	57
Tailing	4,710	70.18	1.40	22.34	3.80	30.31						
No. 2 sludge, 95.0 per cent minus 40 mesh:												
Table feed	28,195	100.00	7.90	100.00	4.70	100.00	26.00	100.00				
Sulphide concentrate	14,195	50.40	15.10	96.23	7.40	79.83			2:1	96.23	15.10	37
Tailing	14,000	49.60	.60	3.77	1.90	20.17	30.25	57.77				
No. 3 sludge, all minus 40 mesh and 10 per cent minus 200 mesh:												
Table feed	4,168	100.00	6.40	100.00	4.30	100.00						
Sulphide concentrate	1,168	28.00	20.80	91.00	11.60	75.80			3.6:1	91.00	20.80	54
Tailing	3,000	72.00	.80	9.00	1.45	24.20						
No. 4 sludge, 65 per cent plus 120 mesh:												
Table feed	4,168	100.00	6.40	100.00	4.30	100.00						
Sulphide concentrate	935	22.40	24.40	85.89	13.80	72.60			4.4:1	85.89	24.40	64
Tailing	3,233	77.60	1.20	14.11	1.50	27.40						
No. 5 sludge, about same as No. 4:												
Table feed	11,500	100.00	6.30	100.00	4.20	100.00						
Sulphide concentrate	2,280	19.74	27.00	84.75					5:1	84.75	27.00	
Tailing	9,220	80.26	1.20	15.25								
No. 6 sludge, all minus 40 mesh:												
Table feed	15,419	100.00	5.20	100.00	3.50	100.00	28.00	100.00				
Sulphide concentrate	3,819	24.77	19.30	92.05	10.50	74.22	16.00	14.02	4:1	92.05	19.30	50
Tailing	11,600	75.23	.55	7.95	1.20	25.78	32.00	85.98				
No. 7 sludge, practically all minus 100 mesh:												
Table feed	10,000	100.00	5.08	100.00	10.84	100.00		100.00				
Sulphide concentrate	3,000	30.00	14.60	86.22	26.23	72.60	5.74		3.3:1	86.22	14.60	74
Tailing	7,000	70.00	1.00	13.78	4.25	27.40						

TABLING ON A QUARTER-SIZE DEISTER-OVERSTROM.

The tests on the small table were followed by a number of tests on a quarter-size Deister-Overstrom slime table. A few that are representative will be described. The following test offers much valuable information. Table 6 gives the sizing analysis of the table feed.

TABLE 6.—*Sizing analysis of a sludge.*

Mesh.	Per cent of weight.	Accumulative per cent of weight.
Minus 20, plus 40	2.8	2.8
Minus 40, plus 80	50.8	53.6
Minus 80, plus 120	21.4	75.0
Minus 120, plus 200	14.0	89.0
Minus 200	11.0	....
Minus 20 mesh	100.0	100.0

This sludge is coarser than some of those shown in Table 5; 75 per cent of it is plus 120 mesh, and only 11 per cent is minus 200 mesh. The small proportion that is plus 40 mesh does not prevent the sludge from being easily tabled. Thus the writers found that an unclassified minus 20-mesh feed could be tabled successfully, except when it contained considerable material that was minus 20 but plus 40 mesh. When there is much coarse gangue, it washes into the zinc band and contaminates that badly.

TABLE 7.—*Tabling of sludge on quarter-size Deister-Overstrom table.*

Product.	Per cent of weight.	Zinc.		Iron.	
		Assay, per cent Zn.	Per cent of feed content.	Assay, per cent Fe.	Per cent of feed content.
Feed	100.00	4.65	100.00	3.9	100.00
Sulphide zinc concentrate	22.65	19.50	94.50	8.2	47.30
Iron concentrate	1.69	1.80	.65	40.4	17.50
Table tailing	75.66	.30	4.85	1.8	35.20
Totals	100.00	....	100.00	....	100.00

Table 7 gives further information on this sample. The zinc concentrate has about the desirable grade of 20 per cent, and the table recovery is high. Other tests with the same sludge, in an effort to get concentrates of higher grade, yielded concentrates assaying as high as 32.4 per cent Zn, but the recovery was lower, about 86 per cent, and the tailing assayed 0.9 per cent Zn.

The distribution of the iron in this test shows that although the iron content was lower than in most sludges, it was possible to cut out on the table an iron concentrate assaying more than 40 per cent

Fe (87 per cent marcasite) and less than 2 per cent Zn; the loss of zinc is small and the iron (sulphide) would be available for sulphuric acid manufacture. Furthermore, the iron content of the table tailing is nearly one-half that of the sludge feed, which shows that proportionately far more iron than zinc is lost in the tailing. Much slimy marcasite washes across the table into the tailing; also some iron is present in the gangue.

The feed was given only one pass over the table. Making a zinc-limestone middling to be returned would enable one to get a better grade of concentrate with less careful attention to the cutting of the zinc band.

ATTEMPT TO MAKE A 40 PER CENT ZINC CONCENTRATE BY TABLING.

For a time it was thought that by tabling sludges low in iron a medium-grade zinc concentrate, commercially salable, could be made. Sometimes the mining companies can sell concentrates assaying 40 per cent Zn provided the lime (CaO) content does not exceed 3 to 4 per cent. In jigging, one of the governing factors is to keep the lime content in the concentrates down to about 2.5 per cent. The limestone, although of lower specific gravity than blende, is far more difficult to remove by jigging than, for instance, the flint in the ores of the Tri-State field. It was, therefore, found that lime was difficult to remove by tabling.

A number of tests were run with a sludge assaying about 6 per cent Zn and 4 per cent Fe. In one test the material was first tabled to make a clean tailing; then the enriched sludge was again tabled, making an iron concentrate, a zinc concentrate, and a zinc-lime middling which was to be returned to the head of the rougher table. The middling assayed 32.4 per cent Zn and the concentrate only 30.4 per cent Zn, because the coarser zinc, with little or no iron, went into the middling; the zinc concentrate had to contend with iron as well as lime, whereas the middling had to contend with lime only. The lime content of the concentrate was too high, 5.4 per cent CaO.

A simple classification was also tried. First the sludge (all minus 40 mesh) was tabled to make a clean tailing; then the concentrate was screened on 80 mesh and the two resulting products were tabled separately in an effort to obtain a satisfactory zinc concentrate. Although the zinc concentrate assayed as much as 35 per cent zinc, its lime content was still 5 to 6 per cent.

The writers believe that with careful manipulation it may be possible ocasionally to make a 40 per cent zinc concentrate low in lime, but, in general, this result is not feasible because of the expense and care involved and because a much higher recovery is effected by making a 20 to 25 per cent zinc concentrate and sending it to a flotation plant for a final grading up. In general, such low-grade concentrates made for flotation will contain 5 to 9 per cent CaO.

TABLING ON A QUARTER-SIZE BUTCHART.

After the work had progressed several months, Mr. W. A. Butchart, of Denver, kindly gave to the Wisconsin mining school a quarter-size 3 by 8 foot Butchart concentrating table, a model much used in the Tri-State field. This table was used by the writers in many tests. Mr. J. H. Carney, of the Butchart company, spent about 15 days with the writers tabling sludge and fine tailings; the speed, length of stroke, riffling, etc., were changed in order to ascertain the effect and to determine the most suitable combination of these factors for the sludge to be treated.

Sludge and fine tailings (minus 20 mesh) mixed in the ratio of 1:2 and unclassified were tabled with rather poor results, the coarse gangue going into the zinc band and the wide gradation of sizes preventing the formation of clean bands of mineral, the several bands blending too much. In consequence a small launder classifier, like those used in the Tri-State field, was built for a two-way classification, the coarse material entering the underflow and the fine material overflowing into the launder.

The material was fed to the table with a scoop, and a middling—that is, a product between the zinc band and the tailing—was caught and returned to the head of the table every two or three minutes. This circulating load was estimated to be about 10 per cent of the total table feed. Samples for analysis were caught while the table was running.

Table 8 gives some of the results obtained on the Butchart table after the mixed tailings and sludge had been separated by classifier into two products, designated as coarse sand and fine sand or slime.

The tests proved that an efficient two-way classification would be required to permit hydraulicking the fine iron to the fine-sand table for removal. In this classification the iron was decidedly segregated in the fine sand, thus making the operation of the coarse-sand table easier and increasing its capacity.

The classifier used in the laboratory sent about 85 per cent of the mixed minus 20-mesh jig sand and sludge to the first classification (coarse sand) and 15 per cent to the second classification (fine sand). The amount of concentrate produced on the coarse table would be at least three times as much and the amount of tailing about four times as much as on the fine-sand tables. This point should be remembered by one estimating the average grade of concentrate and of tailing made.

TABLE 8.—*Results of treating fine tailings and sludge on a quarter-size Butchart table.*

No.	Material treated.	Feed.		Concentrate.			Tailing, per cent Zn.	Extraction, per cent.	Middling, per cent Zn.	Remarks.
		Per cent Zn.	Per cent Fe.	Per cent Zn.	Per cent Fe.	Per cent CaO.				
1	Coarse sand....	3.9	......	20.4	21.1	7.2	0.8	81.0	4.7	No iron or lead cut out of coarse or fine sand.
2	Fine sand......	3.9	......	14.0	27.8	6.5	.9	84.0	2.3	
3	Sludge.........	5.0	12.4	18.0	22.4	......	1.0	85.0	3.7	Lead and iron cut out.
3	Sludge.........	5.0	12.4	......	38.2	......	......	......	......	5.3 per cent Pb in iron concentrate.
1	Coarse sand....	3.9	......	24.6	......	......	1.2	75.0	2.9	Riffle changed.
4	Fine sand......	5.0	......	13.8	......	......	1.0	86.0	1.2	Not enough iron cut out.
5	Coarse sand....	3.2	......	21.4	......	......	.9	75.0	2.0	70 per cent of feed on 100 mesh.
6	Fine sand......	5.0	......	12.4	......	......	.8	90.0	2.4	No lead or iron cut out.
5	Coarse sand....	3.2	......	22.2	......	......	.8	80.0	3.3	Classified two ways.
6	Fine sand......	5.0	......	17.8	......	......	.4	88.0	3.7	Iron and lead cut out.
6	Fine sand......	5.0	......	......	42.6	......	......	......	......	Iron and lead concentrate.
7	Mixed, fine and coarse.	3.2	......	17.8	17.2	......	.8	80.0	2.5	No lead or iron cut out.
8	Coarse sand....	3.2	......	18.6	......	......	.3	95.0	1.2	Lead and iron cut out.
8	Coarse sand....	3.2	......	32.6	......	......	......	......	......	Cut for clean zinc.
9	Fine sand......	4.0	......	18.6	......	......	.9	80.0	1.2	
9	Fine sand......	4.0	......	......	34.6	......	......	......	......	13.7 per cent Pb in iron concentrate.
10	Sludge.........	5.0	7.2	26.2	......	......	.9	87.0	2.1	60 per cent of feed plus 100 mesh.
10	Sludge.........	5.0	7.2	25.8	......	......	1.0	85.0	1.0	Riffle changed.
8	Coarse sand....	3.2	......	26.8	......	......	.7	82.0	1.6	Table speed changed.

The grade of the commercial concentrate that can be produced and the mineral content of the tailing will vary with the individual mine. However, a zinc concentrate averaging 20 to 25 per cent zinc can readily be made from the ores tested on this table and the average tailing will probably contain about 0.5 per cent zinc. Figuring from the extraction basis, it seems that with a feed containing 3 per cent zinc the extraction will average 80 per cent, with 4 per cent zinc feed the extraction will be above 85 per cent, and with 5 per cent feed the extraction will average more than 90 per cent.

The favorable conditions in a commercial plant can not be duplicated in laboratory tests like these, in which the mechanical details of getting the feed to the table, catching samples, etc., take most of the operator's time with the result that the table is not properly adjusted to each individual feed. Hence, it is entirely fair to assume that a commercial plant will give somewhat better results, for each table would be adjusted to its feed and operated steadily under the most favorable conditions.

With the launders arranged for automatically cutting the tailing, middling, and concentrate a cleaner zinc concentrate could be made, as is shown by one or two "special" concentrates (No. 8, coarse sand) cut as concentrates are taken in the Tri-State field.

In the laboratory, a table speed of about 245 r. p. m. and a stroke of $\frac{5}{8}$ or even $\frac{1}{2}$ inch gave favorable results. The best riffling, as well as the best speed and length of stroke are ascertained most advantageously on a given table with its particular feed in a commercial plant.

AMOUNT OF ZINC CONCENTRATE PRODUCED DAILY BY TABLING.

Table 9 shows the tons of zinc concentrate that might be expected daily from a table at a given mill, when the feed carries 4 per cent Zn, the tailing 0.8 per cent Zn, and the feed is 50 tons. A mill that had to treat daily 50 tons of sludge and fine tailings assaying 4 per cent Zn and keep the zinc in the tailing down to 0.8 per cent would produce 8.3 tons of 20 per cent zinc concentrate.

TABLE 9.—*Amount of zinc concentrate to be expected from certain amounts of feed; assays of feed and of products.*

Zinc assays.			Ratio of concentration.	Amount of concentrate with feed tonnage of—		
Feed.	Tailing.	Concentrate.		60	50	40
Per cent.	*Per cent.*	*Per cent.*		*Tons.*	*Tons.*	*Tons.*
3.0	1.0	20.0	9.5	6.3	5.3	4.2
3.5	1.0	20.0	7.6	7.9	6.6	5.3
4.0	1.0	20.0	6.3	9.5	7.9	6.3
4.5	1.0	20.0	5.4	11.0	9.2	7.4
5.0	1.0	20.0	4.7	12.6	10.5	8.4
3.0	.8	20.0	8.7	6.8	6.0	4.6
3.5	.8	20.0	7.1	8.4	7.0	5.6
4.0	.8	20.0	6.0	10.0	8.3	6.6
4.5	.8	20.0	5.2	11.6	9.6	7.7
5.0	.8	20.0	4.6	13.1	10.9	8.8
3.0	.5	20.0	7.8	7.7	6.4	5.1
3.5	.5	20.0	6.5	9.2	7.7	6.1
4.0	.5	20.0	5.6	10.7	9.0	7.2
4.5	.5	20.0	4.9	12.4	10.4	8.2
5.0	.5	20.0	4.2	14.0	11.7	9.2

In calculating these data, the standard formula for ratio of concentration was used. Ratio of concentration $=\frac{C-T}{H-T}$. Where C is zinc assay of concentrate, H is zinc assay of heads or feed, and T the zinc assay of the tailing. This gives the number of tons of feed required to produce 1 ton of concentrate. If it is found desirable to produce a 25 per cent zinc concentrate, the number of tons of concentrate produced would be about 25 per cent less than shown in the table, allowing for a small loss in table efficiency. To find the tons of concentrate produced from a given feed, the formula becomes: Tons of concentrate=tons feed$\times\frac{H-T}{C-T}$.

The possible daily production of zinc concentrate by tabling would have to be determined in each mill, but some estimates may be made for purposes of information. About 20 per cent of the dirt milled would be fine tailings and sludge; if 250 tons were hoisted daily, 50 tons would be available for tabling. On the assumption that this fine stuff contains 4 per cent Zn and the tailings assay 1 per cent Zn, the daily output of 20 per cent zinc concentrate would be 7.9 tons. (See Table 9.)

AMOUNT OF IRON CONCENTRATE PRODUCED DAILY BY TABLING.

If the preceding figures are to be used for calculating the amount of iron produced, then let us assume that the sludge assays 6 per cent Fe and the recovery of iron in the iron concentrate is 20 per cent. If more iron is recovered here, less may be recovered from the flotation tailing, and vice versa. A 20 per cent recovery would amount to 1,200 pounds of iron daily or 1.5 tons of concentrate (40 per cent Fe) suitable for the manufacture of sulphuric acid.

AMOUNT OF LEAD CONCENTRATE PRODUCED DAILY BY TABLING.

The lead content of these sludges and fine tailings does not run over 1 per cent except rarely. The galena is not excessively slimed as it has passed through only the primary crushers, breaking to $\frac{7}{16}$ inch, and therefore can be recovered readily on the table. Assume that the sludge assays 0.5 per cent Pb and that 50 per cent is recovered. The recovery would be about 250 pounds a day (312.5 pounds of 80 per cent concentrate) or enough to pay a large part of the cost of tabling.

AMOUNT OF LIMESTONE TAILING PRODUCED DAILY BY TABLING.

On the basis of the above assumptions, the limestone tailing would amount to about 40 tons a day. It has value for liming soils, as is shown by comparing the analysis of one of these table tailings with that of the "lime fertilizer" now bought by the mining companies:

Analysis of limestone tailing and "lime fertilizer."

Item.	Zn.	Insol.	Fe.	Pb.	CaO.	Mgo.	S.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Table tailing	0.9	13.7	3.7	0.15	28.8 or 51.4 $CaCO_3$	10.2 or 21.4 $MgCO_3$	3.4
Lime fertilizer	....	7.0	.25	....	27.9 or 49.8 $CaCO_3$	18.8 or 39.5 $MgCO_3$	.1

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In 1921, this "lime fertilizer" was worth $2.20 a ton f. o. b. shipping point. Furthermore, the coarse limestone tailing, plus 20 mesh, is of value in road building. Also, the Wisconsin State Geological Survey is studying the possible use of these tailings as aggregate in concrete.

On the basis of the figures assumed, a table plant would produce daily 7.9 tons of 20 per cent zinc concentrate, 1.5 tons of 40 per cent iron concentrate, 312 pounds of 80 per cent lead concentrate, and 40 tons of limestone tailing.

FLOW SHEET AND EQUIPMENT OF TABLE PLANT.

In present practice the tailing from the jigs flows to a settling box, from which the overflow water and fine solids pass to the sludge pond and the coarser solids to the tailing elevator.

This tailing would probably be screened in a trommel at 20 mesh, the coarse going to the tailing pile and the sludge to an elevator and thence to a fed box or a cone feeding the tables. The sludge would be sent to a settling tank and then join the tailing in the elevator to the feed box. Either a cone or a launder classification might be used and the feed go directly to the tables. Two tables, one for coarse sand and one for fine, would probably handle the tonnage available, but it would be safer to have two coarse-sand tables. The tables would make an iron-lead concentrate, a zinc concentrate, a middling and a clean tailing. The middling, about 10 per cent of the table feed, or a circulating load of 10 per cent, would be returned to the head of the table. A zinc-iron middling might be made for re-treatment. The iron-lead concentrate could be stored and, after a quantity had accumulated, retabled to separate the lead and iron as high-grade products.

Before an installation is made, attention should be given to the use of settling boxes, at the end of the jig, for separating coarse and fine sand. In one mill examined, the sludge contains an exceptionally large amount of coarse material because of the use of a shallow settling box. At two different times the tailing going to the tailing pile was sampled and examined, but each time there was only 10 per cent of minus 20-mesh material present which assayed between 1 and 1.3 per cent zinc. Screening such tailings would not be feasible, but treatment of the sludge would be. This mill may be an exception but it seems as though at some mills a shallow settling box might be the only device necessary for separating the coarse and the fines of the jig tailing before tabling.

A number of revenue-yielding products are obtained from the tables. One, the zinc concentrate, is unfinished and is to be sent to a central flotation plant for further grading up. Curiously enough, products similar to all of those made in the table mill are produced at the flotation plant.

FLOTATION OF LOW-GRADE TABLE CONCENTRATES AND OF JIG CONCENTRATES.

INTRODUCTION.

The low-grade table concentrate that is to be treated by flotation for a final grading up is unique in character and composition. It is granular, free from foul water, clayey material, or excessively slimy material, and comparatively free from oil rock. It contains 60 to 75 per cent sulphides; the remainder is gangue that may contain locked grains of limestone, marcasite, and oil rock. The main problem in its flotation is to separate the marcasite and sphalerite, but no particularly successful method of separating the two by flotation has been known.

During the year 1920 the writers had the privilege of working on a series of complex sulphide ores from Colorado. The laboratory methods applied were flotation and gravity concentration. Before that work was begun, an effort was made to diagnose the Colorado ore-treatment problem in order to standardize the mode of treatment. After some consideration it was decided that the chief requirement in devising a standardized treatment for the Colorado ores was to make a clean zinc concentrate. The word " clean " is used advisedly, for a concentrate might be rich in zinc and yet contain enough gold and silver to make the loss prohibitive. It was evident that if a clean zinc concentrate containing a large proportion of the zinc could be made, the recovery of the gold, silver, lead, and copper in a marketable concentrate would be greatly facilitated. Accordingly a search was made for chemically reactive flotation agents that would aid in floating sphalerite away from the associated minerals. An acidified solution of sodium dichromate proved to have some of the desired qualities, and when flotation was judiciously combined with gravity concentration, some success resulted, the chemicals retarding decidedly the flotation of pyrite and retarding advantageously the flotation of galena, silver minerals, and gold. Interest centered chiefly on the sharp separation of sphalerite and pyrite, as the pyrite was " deadened " and the sphalerite floated freely. The work had to be concluded before it was carried through the laboratory stage.

An examination of the Wisconsin marcasite-sphalerite ores to ascertain whether the marcasite could be deadened as easily as was the Colorado pyrite gave encouraging results, and led to the making of the cooperative agreement between Wisconsin zinc-mine operators and the Bureau of Mines.

FACTORS OF FLOTATION.

In the flotation of any sulphide many factors have to be determined before the best results are obtainable. Some of the factors are the

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chemical reagents and oils used, the water, and the fineness of grinding. When flotation was new it was really a paradox, but now it is rapidly reaching the state of development of a process requiring water preparation in accordance with chemical and physical laws, and ore preparation in accordance with physical laws, the objects being to make a mixture that will yield a froth laden with economic minerals. As a consequence, the science of flotation consists of classified facts relating to the interaction of the prepared water and the prepared ore. Some of the laws are yet vague, but many flotation results have been coordinated with the nature of the prepared water and ore.

The method of procedure adopted for the flotation of the low-grade concentrate obtained by tabling the Wisconsin ores is to " rough " in water with an oil, and then send the rougher overflow to a series of cleaners, four or more, the last yielding a high-grade zinc concentrate.

GRINDING THE TABLE CONCENTRATE FOR FLOTATION.

Grinding the table concentrate for flotation may be considered under two heads—stage grinding and the fineness necessary for flotation. The low-grade concentrate from the table must be ground to prepare it for flotation because some grains are nearly 1 mm. in size. Tube mills would be used for the purpose. So-called stage grinding, wherein material is discharged as soon as it reaches the required fineness, is easily practiced in a plant with modern tube mills, but in a laboratory is difficult and tedious. Excessive grinding or sliming of part of the material is objectionable. Ideal grinding would reduce every grain to the size required for successful flotation and none would be finer. Galena, which is present in small amounts in the flotation feed, is easily slimed, and is almost irrecoverable in tabling middlings or flotation concentrates, if stage grinding is not practiced; when the galena is excessively fine its floating is not entirely retarded by acid-dichromate solution, consequently it contaminates the flotation zinc concentrate. Also, removing a plus 300-mesh grain of marcasite from sphalerite by flotation is far less difficult than removing a minus 300-mesh grain. Many flotation zinc concentrates have been screened on 300 mesh by the writers, and the resulting products analyzed for iron and zinc. In every test the iron was segregated in the finer material and showed greater reluctance to being removed than the coarser iron.

The old idea that excessive grading is advantageous to flotation or that flotation is a "cure-all" for slime must be at least modified if not abandoned. It has been found repeatedly with ores from several districts that it is increasingly difficult to effect separation of sulphides by flotation, as the proportion of excessively fine material increases. Within limits, the coarser the flotation charge, the easier it is to make a separation of sulphides, with or without the use of chemical reagents.

Another objection to excessive grinding is that it releases more and more of the objectionable oil rock from the locked grains of sulphur rock. Furthermore, calcite, one of the gangue constituents, is somewhat floatable and is likely to cause trouble when ground too fine.

Grinding was done in the laboratory either in a small porcelain jar with flint pebbles, or in two larger jars, one porcelain lined and containing flint pebbles, and the other iron lined and containing iron balls. The method of grinding was to load one of the jars with a charge of ore and about twice the weight of water. The small jar took a charge of about 2 or 3 pounds of ore and the larger jars 10 to 15 pounds. After it had been ground a short time, the ore was screened on some particular mesh and the oversize returned to the jar; after further grinding for a short period, the operation was repeated. This procedure approximates grinding with a tube mill in a closed circuit.

The method of grinding frequently effects the floatability of a product. All products discussed in this work were ground wet. The difference in effect between dry and wet grinding on floatability was not determined, as there would be no occasion for dry grinding in the writers' process. Grinding with flint pebbles may give results different from those of grinding with steel balls. Iron rubbed off of the balls is said to have an effect on the flotation of zinc, but in tests of the Wisconsin ores no such effect was noted. Therefore, grinding was done either with flint pebbles or with iron balls.

Flotation of sphalerite may usually be accomplished with ore crushed to pass 80 mesh, but this is not true of Wisconsin blende, the mineral associates of which, particularly marcasite, give a unique mixture. Crushed to 80 mesh this sphalerite floated very slowly, and much of the mineral was left in the tailing, whereas the mineral should float rapidly and leave a low-grade tailing. When the flotation feed was crushed through 80 mesh only, the zinc that did float in the roughing machine was slow to float in the cleaning operations, and less marcasite floated in the roughing than if the charge had been ground finer. The loss of zinc in the flotation tailing should not be overlooked. From the tests made to determine the fineness of grinding necessary the conclusion was reached that the flotation feed should be ground through 120 mesh.

A screen analysis of a flotation tailing from a minus 80-mesh feed, shown in Table 10, verified the conclusion that losses were excessive in the plus 120-mesh part of the tailing. From Table 10 it might seem that crushing finer than 120 mesh should be practiced, but two facts contradict this idea: First, it has been possible in the many tests made in this investigation to obtain as a rule a tailing assaying 1 per cent Zn or less; second, very fine grinding presents dangers that have been described above.

TABLE 10.—*Screen analysis of a flotation tailing from a test in which the feed was crushed through 80 mesh.*

Product.	Per cent of weight.	Zinc.	
		Assay, per cent.	Per cent of total content.
Composite tailing	100.00	2.50	100.00
Minus 80, plus 100	13.0	9.60	49.92
Minus 100, plus 120	14.0	4.20	23.50
Minus 120, plus 160	6.0	1.80	4.30
Minus 160, plus 200	26.0	1.20	12.44
Minus 200	41.0	.60	9.84
Total	100.0	....	100.00

BULK FLOTATION.

It is always a question whether the separation of two sulphides should begin in the roughing machine or should be deferred until cleaning is begun. In so-called differential flotation chemical reagents are usually added to the fresh ore in the rougher machine to retard the flotation of one mineral while another is floating; then further reagents are added to cause the retarded mineral to float. Invariably the procedure results in large tailing losses and in imperfect separation of minerals. On the other hand, bulk flotation, wherein the rougher treatment consists in floating two or more minerals together and making a finished tailing, has much to commend it. Chemicals are rarely if ever used. The quantity of tailing made represents more than half the original bulk. The overflow from the rougher consists primarily in sulphides and in very little gangue. To this product chemical reagents acting primarily on the marcasite may be added in small and definite quantity; the reagents do not expend themselves on a large part of the gangue that can be removed without their use. The cost of reagents is less than when separation begins in the rougher, the quantity of material treated is less, and the tailing loss is smaller.

Another factor in favor of bulk flotation for Wisconsin ores is that zinc blende does not float as readily in the dichromate solution before it has been floated with oil as it does after roughing flotation. Grinding with oil in preparation for flotation can be adopted, but should be done with caution or the overflow will be too low grade. Furthermore, without the presence of chemical reagents much of the marcasite is dropped in the tailing in the rougher.

When two sulphides—such as sphalerite and marcasite—are present in an ore, each has a different rate of flotation; if they are floated in ordinary water with an oil. the sphalerite predominates at the beginning of the overflow, gradually decreasing in quantity as the test progresses. The marcasite on the other hand increases. Toward the end of the test the overflow is almost entirely marcasite, with very little blende present. Hence there may be left in the tailing a quantity of iron that has been removed from the zinc without chemical reagents; thus the burden of the cleaning machine is lessened.

In the flotation tests made by the writers, no chemical reagents were added in roughing, oil alone being used, and an iron-limestone tailing low in zinc was removed from the circuit.

CHEMICAL REAGENTS.

In the work on the Colorado ores[3] an extensive search was made to find reagents that would separate iron and lead sulphides from zinc sulphide. The conclusion reached was that ordinary inorganic salts, such as sulphates and nitrates, do not possess the requisite property of effecting a sharp, definite separation between two sulphides. They tend to separate minerals, but with present knowledge that tendency can not be developed. A reagent or a salt must have pronounced reactivity to effect such a separation, as the solutions of the salt are very dilute. Hence the reagents suitable for these separations are limited to strong oxidizing and reducing reagents. Acid dichromates, which are among the best oxidizing agents known, retard pyrite and marcasite but permit sphalerite to float. A desirable effect of these reagents is that they alter the surface of the mineral. A reagent may act upon a sulphide surface and yet form a compound that is soluble in the medium in which flotation is performed; then the mineral might still float. On the other hand, if the surface is coated with an insoluble compound, the mineral is usually rendered immune to flotation in that medium.

Upon sphalerite acid dichromate has no effect, but usually forms a yellowish compound, possibly lead chromate, on the surface of galena. When a strong solution of acid dichromate has acted on galena, a coating of sulphur has been found on the surface. Just what compound is formed on the galena is uncertain as it depends somewhat on the strength of the oxidizing agent. After the dichromate treatment the surface of galena is lusterless, and has somewhat the color of weathered pyrite. Marcasite is tarnished by the same treatment; the compound formed on its surface is of unknown composition. No attempt was made to name the compound on the surface of iron sulphide. Here is an opportunity for advantageous research to obtain more knowledge of the reaction of the acid dichromate on these three sulphides.

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[3] Coghill, W. H., and Anderson, C. O., Low-grade and complex ores of Colorado: Tech. Paper 283, Bureau of Mines. In press.

Galena consumes instantaneously the oxidizing power of acid dichromate, pyrite or marcasite consumes it gradually, and sphalerite does not consume it at all. Dichromate solutions may be acid, neutral, or alkaline, but the latter two possess no oxidizing power. Patents describe the use of neutral and alkaline dichromates in effecting the separation of these sulphides, but no patent has been found on the use of acid dichromate for this separation, though it is more effective than either alkaline or neutral solutions. In the Colorado work it was determined that the acidity must be carefully controlled. The chemical equivalent of sulphuric acid necessary for the release of the total oxygen of sodium dichromate is 0.81 grams per gram of dichromate. Flotation of lead and iron sulphides is retarded in a solution containing these equivalent quantities, but is retarded most effectively when about 0.2 grams sulphuric acid is used per gram of dichromate. In the work with Wisconsin products a new factor enters that makes the control of acidity less delicate and important: The limestone present in the charge immediately consumes any acid in excess of the requirements of the dichromate. Within limits, an excess of acid would never occur in the flotation of Wisconsin ores; the limestone is attacked so easily by sulphuric acid that it acts as a safeguard against an excess of acid in the flotation medium. Acid should not, however, be put promiscuously into the flotation machine, as products of decomposition may become an influencing factor in flotation.

Sodium dichromate was used because it seems as effective as potassium dichromate and less of it by quantity is required according to chemical equivalents. Sodium dichromate is cheaper also than either the potassium or calcium compounds; the market quotation for it during 1921 averaged close to 8 cents per pound in carload lots. For testing, a 1 per cent solution of sodium dichromate was made; after the addition of dichromate to the flotation medium in the quantities used in this work, its strength was generally 0.02 to 0.05 per cent.

Sulphuric acid was the only acid used, because it is usually employed in oxidizing solutions of dichromates, and because it was cheap, selling then for 1 cent or less per pound.

To the 1 per cent dichromate solution was added 0.5 c. c. or 0.9 gram sulphuric acid per gram of dichromate, practically the chemical equivalent of acid necessary to release the total oxidizing power of the dichromate. Smaller quantities of acid did not remove marcasite as effectively in the cleaning operations. Table 11, which records results of using different degrees of acidity, shows cleaner zinc prod-

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ucts with increased quantities of acid. The results were obtained by one roughing and one cleaning of each charge.

TABLE 11.—*Results of cleaning zinc products in dichromate solutions with different amounts of acid.*

Product.	Per cent of total weight.	Zinc.		Iron assay, per cent.	Amount of acid per gram of dichromate, c. c.
		Assay, per cent.	Per cent of feed content.		
Feed	100.0	25.4	100.0	....	
Zinc overflow	37.4	55.4	81.5	5.4	Neutral, no acid.
Zinc-iron underflow	25.7	16.6	16.7	24.4	
Tailing	36.9	1.2	1.8	....	
Zinc overflow	33.0	58.4	77.0	3.4	0.25.
Zinc-iron underflow	27.0	19.6	21.1	23.0	
Tailing	40.0	1.2	1.9	....	
Zinc overflow	31.8	60.6	77.0	2.8	0.50.
Zinc-iron underflow	29.5	18.0	21.1	24.4	
Tailing	38.7	1.2	1.9	....	

In making these tests a table concentrate assaying 25.4 per cent zinc was stage-crushed through 120 mesh, and about 1,000 grams was floated (roughed) in water with No. 400 Pensacola Tar & Turpentine oil. In each test a tailing was made. The overflow was cleaned in a solution of 1 gram sodium dichromate in about 3 liters of water, and the quantities of acid indicated in the last column of Table 11. A zinc overflow and a zinc-iron underflow resulted.

With increasing acidity, the overflow was higher in zinc and lower in iron. No particular advantage was observed in increasing the acidity above 0.5 c.c. per gram of dichromate. Some acid was probably consumed in neutralizing the lime. Table 11 contains proof that the acidified dichromate is more effective than the neutral. Dichromate solutions made alkaline with sodium carbonate or hydroxide are even less effective than neutral solutions. Furthermore, clotted particles of blende, discussed later, cause much trouble in alkaline solutions.

EXPERIMENTS WITH COPPER SULPHATE.

As it promotes the flotation of sphalerite, copper sulphate is used extensively in the Tennessee and Tri-State districts. The late Mr. Roessler reported in his work for the Vinegar Hill Zinc Co. that copper sulphate aids the flotation of blende and inhibits the flotation of iron sulphide. The Colorado tests mentioned gave the impression that copper sulphate and dichromate did not work advantageously in the same system. Several tests, however, were made employing copper sulphate in the roughing treatment, with the idea that if the roughing in copper sulphate was successful, a cleaning in dichromate would follow. Table 12 gives results of comparative runs with varying amounts of copper sulphate.

TABLE 12.—*Comparative runs (roughing flotation) with stated amounts of copper sulphate.*

Product.	Per cent of total weight.	Zinc.		Amount of $CuSO_4$ grams.
		Assay, per cent.	Per cent of feed content.	
Zinc overflow	45.6	52.4	90.1	None.
Tailing	54.4	4.8	9.9	Do.
Zinc overflow	42.0	55.6	88.9	0.5
Tailing	58.0	5.0	11.1	5
Zinc overflow	....	(1)	....	1.0
Tailing	....	(2)	....	

[1] Nothing floated.

[2] Flotation completely inhibited.

These tests were made on a table concentrate stage-crushed through 120 mesh as above. About 1,000 grams was employed in 4 liters of water in each test; the oil was Cleveland Cliffs No. 2 hardwood creosote oil, which is much used with copper sulphate in the Tri-State district. With a small quantity of copper sulphate—0.5 gram—the grade of zinc overflow was improved. But when the amount was increased to 1 gram, nothing floated, regardless of the quantity of oil added. Not only was marcasite inhibited from floating, but the sphalerite also. The test was repeated, using No. 400 Pensacola Tar & Turpentine oil (see p. 37) instead of Cleveland Cliffs No. 2 oil. A zinc overflow assaying 45.6 per cent Zn and a tailing assaying 1.3 per cent Zn were made, indicating that No. 400 oil acted more effectively with copper sulphate than did No. 2 oil. Furthermore, copper sulphate with No. 400 oil improved the rate of flotation; it hindered the flotation of iron sulphide, possibly because it increased the rate of flotation of zinc sulphide.

For some time it had been believed that copper sulphate acted as a water-treatment reagent, thus improving indirectly the flotation of sphalerite. The old belief that a film of copper deposited on the surface of blende promoted its flotation did not seem to the writers to explain fully the facts. The copper sulphate, however, probably neutralized certain salts dissolved in the water. To obtain data in support of this belief, two tests were performed similar to those in Table 12, distilled water being employed in place of the city water of Platteville. In the first test, Cleveland Cliffs No. 2 oil with no copper sulphate was used. Very free flotation of both the zinc and the iron sulphides was observed, the sulphides floating far more readily in distilled water than in city water. In the second test, the same oil being used with 0.5 gram copper sulphate, no sulphides floated; the froth, which was very thin, was barren, whereas in the second test of Table 12 the flotation was good. In all tests the ore fed to the flotation machine had been tabled and consequently washed free of soluble constituents that might have contaminated the water in the flotation cell.

These tests seemed to prove conclusively that copper sulphate is a water-treatment reagent. In test 2 of Table 12 the city water contained various salts which no doubt the copper compound neutralized. When distilled water, in which these salts were absent, was used, the copper sulphate could not react on them, but it did have a detrimental effect on the flotation of sphalerite. Copper sulphate in distilled water gave a clear solution but in city water gave a precipitate. In the last test of Table 12 copper sulphate was apparently present in excess; the salts in the water were neutralized, but there was still some sulphate present to affect the flotation of the sulphides. Similarly may be explained the difference in the action of the two oils used with the same amounts of copper sulphate. Cleveland Cliffs No. 2 oil is acid and does not contain constituents on which the copper sulphate would react and expend itself. No. 400 Pensacola Tar & Turpentine oil, on the other hand, contains free alkali upon which copper sulphate, being acid, would react.

A series of tests was made with amounts of copper sulphate less than that known to be deleterious. The results showed that copper sulphate did not cause any decided difference in the grade of the overflow or the tailing. It seemed to have little effect on the amount of the iron, lime, and insoluble that floated, or on the rate of flotation, or the time required for the completion of a test.

In the opinion of the authors the results with copper sulphate do not warrant its use in roughing. Copper sulphate was not used in the final tests covered in this report, nevertheless the above discussion presents a different point of view on the action of copper sulphate in the flotation of sphalerite.

WATER.

The deep-well water of the Wisconsin zinc district flows through many limestone formations and is alkaline. The pumping station of Platteville supplied the following analyses of the water:

Analyses of water from deep well at Platteville, Wis.

Analysis by Economy Laboratories, Kansas City, Mo., October, 1919.

Item.	Well water. *Grains per gal.*	Creek water. *Grains per gal.*
Organic	2.00	5.50
Silica	1.80	1.96
Iron oxide	0.90	0.35
Sodium chloride	3.51	2.80
Sodium sulphate	12.59	12.35
Magnesium sulphate	2.11	4.18
Calcium carbonate	12.14	17.50
Magnesium carbonate	7.63	7.23
	42.68	51.87

[42.68 grains per gallon equals 0.73 gram per liter.]

Analysis by the Chicago, Milwaukee & St. Paul Railway Co. October, 1918.

Item.	*Grains per gal.*
Oxides	Undetermined.
Calcium carbonate	3.88
Calcium sulphate	15.23
Magnesium carbonate	12.18
Magnesium chloride	1.11
Incrusting solids	32.40
Nonincrusting solids	1.06
	33.46

Analysis by the State chemist.

Item.	*Parts per million.*
Total solids	836 to 730
Hardness	543 to 457
Alkalinity	310 to 308

The total inorganic solids present in the water used in the laboratory were determined by evaporating 10 liters of the water to dryness on a sand bath. The pure white residue amounted to 0.412 gram per liter or 412 parts per million; it was not analyzed, but its composition may be approximated from the above analyses. To neutralize the alkaline constituents (give a solution neutral to methyl orange) requires 0.28 gram of sulphuric acid per liter of water. Acid added in the dichromate solution in the flotation machine probably releases the oxidizing power of the dichromate, neutralizes the alkaline constituents of the water, and then, if still in excess, acts on the limestone.

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Sphalerite as a rule floats best in neutral or acid solutions. This fact and the alkalinity of the water explain in part the relative slowness with which Wisconsin sphalerite floats. In research for the Vinegar Hill Zinc Co., Mr. Roessler found that the addition of ammonium oxalate or phosphate or their equivalents to precipitate calcium and magnesium salts promotes the flotation of blende. These salts were not tested in the present study, but experience with copper sulphate and sulphuric acid tends to confirm Roessler's findings. Sulphuric acid performed much the same function as the above ammonium compounds, that is, it neutralized the calcium and magnesium salts, but did not precipitate them. Much difficulty arose in this work because the overflow was badly clotted and could not be readily cleaned.

CLOTTING.

Clotting is another name for coagulation. When solids are suspended or agitated in a solution, many substances have the property of causing the suspended particles to coagulate or to clot together; other substances have the effect of dispersing or breaking up the floccules. Clotting of the blende was not observed in the roughing, but began to appear in cleaning; it became more decided as the overflow passed to succeeding cleaners, and finally had the consistence of large curds of milk. The cleaning operation then became useless; the clots entrapped the marcasite and gangue, hence the overflow was barely richer in zinc than the underflow. In the flotation machine nearly everything floated in spite of the extremely low density of the pulp at the end of the run. The overflow was fluffy and the solids did not settle. Increased amounts of dichromate did not materially relieve this condition.

Several substances were tried in order to determine which would produce clotting or curdling of the blende, and which would break up or disperse the clots. Alkaline compounds—such as caustic soda—caused intense clotting. Acids—such as sulphuric acid—dispersed the clots and allowed the zinc to float in a granular condition; the overflow settled rapidly and the clear liquid could be readily decanted. Further tests showed that clotting could be easily controlled as follows: The addition of caustic soda to a charge in the flotation cleaner machine produced coagulation; the addition of acid then broke up the clots and small granules resulted. When alkali was added again, coagulation again occurred. Conditions could be reversed as many times as desired while the machine was running.

It was determined that in order to insure best results acid should be added to each cleaner cell, the amount not to exceed an acidity of 0.04 per cent by weight, provided the water was neutral when the acid was added. About 0.15 c. c.—0.28 gram per liter—was re-

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quired to make the water neutral to methyl orange. As limestone consumes excess acid, the flotation solution did not possess acid when titrated. Acid was added to the cleaners just before the addition of dichromate solution, so that the flotation medium would insure the most effective action of the oxidizing agent on the charge of ore.

In the search to remedy clotting no attention was given to the composition of the oil used; later, however, in the study of oils at the Forest Products Laboratory, it was concluded that the oil—No. 400, Pensacola Tar & Turpentine Co.—was in a great measure responsible for the clotting. This point will be discussed later. (See p. 39.) The tests indicated that coagulation is promoted and produced by the alkalinity both of the water and of the oil.

At one time it was thought that grinding with iron balls might produce coagulation, whereas grinding with flint pebbles would not do so. Clotting resulted in the cleaning operation, however, regardless of the mode of grinding.

SOOTY IRON.

In the roughing flotation of some products, a peculiar condition was encountered which was attributed to what may be termed "sooty iron." When a product from a certain mill was crushed through 80 mesh and floated, it produced a black, sooty scum on the froth. Flotation was slow and the sphalerite did not float until this sooty material—which was slimed marcasite and possibly some slimed lead—had come off the machine; also, the volume of the overflow water was unusually large. After this material had floated, the zinc floated freely.

Various expedients were tried. Tabling the flotation feed, in order to remove the slimy marcasite in a side wash from the table, resulted in a product that was readily treated by flotation, but the loss by tabling was excessive; the slime contained as much zinc as iron. The difficulty was overcome most effectively by adding the oil more rapidly and in larger quantities than usual at the outset of the run, thus hastening the flotation of the sphalerite. After the amount of overflow water was cut to normal, the rougher overflow was cleaned readily. Careful stage grinding lessened the proportion of slimy marcasite. Many or most of the products treated by flotation did not contain this sooty iron. It was produced by grinding certain jig concentrates; iron concentrates, minus 80-mesh screen, though ground excessively, did not exhibit this characteristic when floated.

ACIDITY.

In studying the use of acid in roughing, comparative tests were made with city water, also with city water to which enough acid had been added to make it neutral to methyl orange. No particular preference was observed, as the grade of the overflow and the grade of the tailing were about the same, and the rate of flotation was much the same. No effort was made to float the minerals in an acidified solution; to get such a condition, indeed, would be practically impossible; the acid would be consumed by the limestone until all dissolved. All roughing tests herein reported were made with city water without chemical reagents.

Much attention was directed in the Colorado work to the effect of the acidity of dichromate solutions on the flotation of galena. In alkaline, neutral, and acid solutions, galena was rendered somewhat immune to flotation, but was retarded most effectively in solutions that were slightly acid. In the many tests with Wisconsin products, the acidity of the solution was varied in the cleaning treatment. Charges to which a quantity of ground galena had been added showed no noticeable difference in the retardation of this sulphide. Within limits, the amount of acid added per gram of dichromate did not change the degree of retardation, and any excess of acid over a certain amount was consumed by the limestone.

In the Colorado work, any acid present over the small amount necessary to allow the dichromate to act on the surface of the galena dissolved the compound formed on the surface of the galena and again put the mineral in condition for flotation. Sulphuric acid was added in the flotation of Wisconsin products for three purposes: (1) To neutralize the alkalinity of the water and the oil, (2) to prevent clotting, and (3) to release the oxidizing power of the dichromate.

OIL ROCK.

If the oil rock, which is one of the gangue constituents of the Wisconsin ore, is free or unlocked from other minerals, it is readily removed in the tabling preparatory to flotation. In some ores, however, the oil rock is intimately associated with limestone and marcasite, hence some of it is carried into the table concentrate that serves as the flotation feed. This product would carry about 1 to 5 per cent of insoluble containing oil rock, or enough to make trouble in flotation. When a table concentrate is ground for flotation, much of the oil rock is released. It is floatable, probably because of its containing small amounts of oily substances.

Table concentrates from some of the samples tested early in the present investigation did not contain oil rock. For these a so-called counter-current scheme (see p. 41) of grinding up and cleaning was devised that was successful. Later, however, this method failed when tried on a sample from one of the mills in the northern part of the district—where much locked oil rock occurs. When fed to the cleaners the overflow from the rougher machine gave a concentrate very little richer than the feed; sometimes the middlings from the cleaning machines were richer in zinc than was the final concentrate. For some time the difficulty was thought to be due to the marcasite being of a different type—botryoidal—from that in the other ores tested, and hence floating more easily in a dichromate solution. Data, however, could not be obtained to make this assumption tenable. Then oil rock was considered responsible for the unsatisfactory concentrate. Numerous tests were made and much time was spent in proving this assumption, and in remedying the condition. In the tests where no cleaning action seemed to be taking place, the iron floated freely. More dichromate was added to give the marcasite a severer treatment and thus remove it, but without success.

A consideration of the roughing treatment of this material indicated the cleaning could be improved. In roughing with no dichromate present the overflow was richer in zinc than the original feed, and the tailings contained much iron and oil rock. It was decided to keep dichromate out of the first cleaning machines and reserve it for the last cleaners. This procedure was successful, and grading up was done in the first cleaners where the material was treated in water to which a little acid had been added to prevent clotting. The middlings from these cleaners carried oil rock and iron which could be returned to the rougher. In the last cleanings—dichromate added—more iron was removed, and there was little trouble with oil rock. The bulk of the oil rock was removed by water cleaning. It is very likely that any flotation concentrate made from ores from the northern part of the district will carry some oil rock.

The action of oil rock in keeping the dichromate from acting on marcasite may be explained as follows: The oil rock is partly organic, and as oxidizing agents attack organic material the dichromate probably expended itself on the oil rock and became useless for retarding the marcasite.

A rough method of determining the proportion of oil rock in these flotation concentrates is to treat a sample of about 10 grams with aqua regia, filter, weigh the reddish brown residue, and then ignite it. The weight of the white ignited residue subtracted from the weight of the original residue gives the weight of the organic matter in the oil rock.

OILS USED IN FLOTATION.

Many of the oils used in flotation have the same characteristics with respect to floating a given mineral. In general the lighter, thinner oils, free of tarry constituents, are not well adapted to the flotation of sphalerite. The heavy oils, containing more tarry constituents, are better. In the flotation of Wisconsin sphalerite restric-

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tions are imposed by the associated minerals, as well as by the demands of the market, so that an oil successful in other zinc districts probably could not be used. Moreover, to be the most effective the oil must be able to carry the sphalerite through the cleaning treatments without the addition of more oil after roughing, as some of the marcasite rendered immune to flotation by the dichromate would be likely to float again.

Steam-distilled pine oil, for example, was entirely unsuccessful in floating, in city water, sphalerite from ore ground to pass 120 mesh, and a so-called X-Y mixture, now finding favor in the Tri-State field was not effective. The reason for the failure of these oils was probably the same, as both are pine-wood products. In the distillation of pine-wood products, turpentine is obtained at temperatures between 155° and 180° C.; pine oil between 180° and 220° C. One of the products of this distillation is a mixture of turpentine and pine oil that can not be separated into the constituents; this is believed to be the composition of the X-Y mixture.

The oils used extensively in the Tri-State district—No. 2 Cleveland Cliffs hardwood creosote and No. 22 General Naval Stores coal-tar creosote—did not meet requirements. In tests on Colorado ores, No. 2 Cleveland Cliffs and No. 350 Pensacola Tar and Turpentine were used individually with considerable success both in roughing and cleaning. Neither of them, however, would give a low enough tailing with the Wisconsin ores, nor would they carry the sphalerite through the cleaning without the further addition of oil. No. 400 Pensacola Tar and Turpentine oil answered the requirements most closely. With it a tailing low enough in zinc could be made, a considerable amount of the iron did not float in the roughing, and the rougher overflow to a desirable degree in the cleaner machine could be graded up readily.

For various reasons it was desirable to have available another oil that would function as well as No. 400, as the supply of that was limited, and in commercial work it is better to have a choice of at least two oils. Hence, an arrangement was made between the Forest Products Laboratory and the Bureau of Mines whereby one of the authors of this paper spent about a month at Madison in studying and preparing oils. As the study came near the end of the work on Wisconsin products, the data gained there could not be tested fully before work stopped. Information on oils may, however, be gained readily after a small flotation plant has been installed.

The Pensacola Tar & Turpentine Co. describes No. 400 oil under the heading "No. 400 Wood Creosote Oil," as follows:

> The name is really a misnomer, for the oil is a mixture of pine oil and wood creosote oil, and is recovered from the caustic soda still bottoms of the refinery.

We do more or less refined turpentine business, and use caustic soda to remove any saponifiable oils in refining. This combines with a small amount of wood creosote oil entrained in the vapors of the roughing-off still. To recover these the soda is washed out with our waste pyroligneous acid, neutralizing the soda and permitting the oils to float off.

The No. 350 oil of this company is the crude product obtained from the condensers; it contains about 6 per cent pitch.

A comparison of the composition of No. 350 and No. 400 oils showed that their constituents were about the same, except for the presence in No. 400 of some pine-wood "dissolved tar." From oil No. 350 the pyroligneous acid, containing the dissolved tar, has been separated by gravity. In the production of No. 400 pyroligneous acid is added to neutralize the caustic soda; the dissolved tar is precipitated and remains in the oil. Here appeared an opportunity for modifying No. 350 oil, of which there is an ample supply, so that it would be effective for the flotation of Wisconsin sphalerite. As no pine-wood or softwood pyroligneous acid was readily available, some dissolved tar was obtained from hardwood pyroligneous acid.

In any method of distilling hardwoods, four crude products are made: (1) Charcoal; (2) noncondensible gas; (3) settled tar in the liquid condensate; and (4) an aqueous liquid, known as "pyro" acid, which when redistilled leaves a residue known as dissolved tar. This tar is nonvolatile and is 50 per cent soluble in water, but its exact composition is unknown.

Dissolved tar (10 per cent by weight) was added to No. 350 oil and was used in some flotation tests. In a locked test of four runs, made with a product stage ground to pass 120 mesh, the flotation was slower than with No. 400 oil. The froth was not quite as bulky as with No. 400, but became gummy toward the end of each roughing treatment. About 35 per cent more oil was required per unit of ore treated. The rougher tailing assayed 0.80 per cent Zn, and the grade of concentrate was more than 60 per cent Zn. Lead can be floated more freely with No. 350 oil than with No. 400. This combination of No. 350 oil and hardwood dissolved tar would hardly be satisfactory, but might prove efficient if part of it was replaced by No. 400.

A similar test was made with a combination of No. 2 Cleveland Cliffs and hardwood dissolved tar (10 per cent by weight). The flotation feed (minus 120 mesh) assayed 22.2 per cent zinc. This combination of oils floated the sphalerite in a froth small in volume and heavily laden with mineral. Flotation was rather slow as compared with No. 400. The principal objection, however, was that a low-grade tailing could not be obtained. In this test the tailing assayed 5.7 per cent Zn, and the extraction was only 81 per cent, as compared with 95 per cent obtainable with No. 400.

This combination, however, seemed to have certain advantages, as follows: Less iron floated in the roughing treatment and hence less dichromate (only 1 pound per ton) was used in the cleaning treatments, which were also fewer than with No. 400; a desirable grade of concentrate was more easily obtained; and no clotting appeared in the cleaning machines, although no acid had been added to prevent it, the only acid added being that in the dichromate solution. Oil No. 2 is slightly acid and showed no tendency to cause clotting.

If some No. 400 oil were added near the end of the roughing treatment, a low tailing would be obtained. This combination of oils should prove highly desirable. Hardwood dissolved tar, moreover, is very cheap—about 3 cents per gallon—its main use being as a fuel at plants for hardwood distillation. In 1917 about 9,000,000 gallons was produced in the United States.

It is true that No. 400 does not contain some of the lighter fractions found in No. 350, for the boiling-point range of No. 400 is above 185° C.; these lighter fractions, however, are of little or no value in the flotation of Wisconsin sphalerite. An alkalinity test of No. 400 oil showed 0.29 per cent by weight of free sodium hydroxide. In the production of this oil no particular care is used to remove all the caustic soda, hence the tendency of the oil to cause clotting. Some No. 400 oil was washed with dilute sulphuric acid to free it of alkali. The washed oil was used in a weighing flotation test, and proved as effective as the original oil. In the cleaning treatments much less acid was required to prevent clotting. It would probably be best to neutralize the alkali in this oil before using. If the reasons were pointed out to the producer he might be induced to remove the alkali.

Oil No. 400 was used in all tests reported; it was not used in combination with other oils, and was not modified in any of the tests. Further work on oils might be done, but this oil should give satisfaction in any installation. Numerous oils would undoubtedly be tried in a commercial plant.

Hardwood oils should be given consideration in any further study of oils for flotation of Wisconsin sphalerite.

LOCKED TESTING.

Much laboratory testing is done in a minimum of time and the results obtained can not well be translated into milling data. In laboratories a representative method of performing a flotation test is by placing in the machines a small charge of ore, and then making a tailing and an overflow; a concentrate is next made by giving this overflow a cleaning and from weights and figures obtained mill recoveries and grades of products are calculated. Thus the final distribution of the mineral in the middlings is figured out easily.

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Now, the behavior of middlings is one of the most disturbing features of laboratory testing. In almost any physical concentration process middlings are produced; the mineral in them is there for one of the following reasons: (1) Because of incompleteness of separation in a particular step of the process; (2) because of the size of the mineral; (3) because of its configuration; or (4) because of the association of its grains. If the mineral is in the middlings from incompleteness of separation, it will probably be recovered in the concentrate obtained after returning the middlings to the circuit. But if there for any of the other reasons it may, when returned to the circuit, find its way into the tailing and thus increase losses, or it may reach the concentrate and lower its grade. Before accurate figures can be obtained, it is evident that middlings must be allowed to have their effect upon the other products in the test.

The authors advanced one step beyond the usual method of handling middlings in the laboratory by using the system of lock testing; a flotation test by this method was made about as follows: Suppose that the treatment requires one roughing and three cleanings. Let the amount of ore be 10 pounds and be divided into four lots. The first charge or lot of ore is roughed, and an overflow and a finished tailing are made. The rougher overflow is sent to the first cleaning, an overflow and an underflow or first middling being made. The first cleaner overflow is sent to the second cleaning, resulting in an overflow and a second middling or underflow. The second cleaner overflow then goes to the third cleaning, which gives a final concentrate and a third middling. Now, the second charge is to be treated in the same manner, except that the three middlings are to be returned to the circuit of the second charge. The middlings may be handled in different ways. They may be dewatered, combined, ground finer, and then returned to the roughing of the second charge. Or, they may be returned individually, with or without grinding or tabling, to the treatment just preceding the one from which they resulted; thus the second middling from the first charge may be returned to the first cleaning of the second charge.

The middlings resulting from the treatment of the second charge are sent to the circuit of the third charge and so on for any number of charges that may be treated. By this system, the mineral in the middlings can become distributed between tailings and concentrates, so that the grade of products resulting, say, from the treatment of the third charge is comparable to those expected in a mill run where middlings are continuously taken care of. Almost invariably the concentrate from the first charge in a test will assay somewhat higher in zinc than concentrates from succeeding charges; there is no way of sending middlings to the treatment given the first charge. The grade of products taken from the third charge usually indicates the average grade that can be maintained. Tests have been made in which as many as 24 charges of ore have been treated, and the results have shown that the grade of products made in about the third charge may be maintained throughout. To make these tests, two machines were kept running by two operators throughout the day. This locked testing is tedious but where middlings are given consideration it is imperative.

In the final tests reported, four charges of ore or flotation feed were run in each. It is believed that safe conclusions as to the grade of products can be drawn by noting the results from the last two charges.

COUNTERCURRENT CLEANING.

Although the plan of countercurrent cleaning was developed in the early part of this investigation and later abandoned on account of difficulties, yet it is discussed here because it doubtless is new in flotation work, though well known in other branches of metallurgy.

As previously mentioned, the scheme followed comprised a roughing treatment and several cleaning treatments. Countercurrent cleaning has reference to the way the reagent—acid dichromate—was added to the cleaning machines; the strongest solution was added to the circuit where the zinc product was freest of marcasite and other impurities; and the dichromate solution, as it became spent, encountered zinc products of progressively lower grade.

Countercurrent cleaning was operated as follows: The middling from each cleaner returned to the head of the cleaner just preceding it; impurities thus dropped into the underflow of the last cleaning, advanced one step, for each charge treated, toward the rougher machine, where they passed into the tailing. After this method was under way the water with each middling constituted the bulk of the flotation medium in each cleaner. The third middling, with its solution from the treatment of the second charge, thus passed back into the second cleaner and joined the first overflow of the third charge. There was no addition of water to the circuit, except in the rougher to replace that passing out with the tailing, and in the last cleaner to replace that passing out with the concentrate. Dichromate was added in the last one or two of the cleaners, and thus encountered the zinc when it was approaching the concentrate stage and removed the impurities as completely as possible. The partly spent dichromate then passed backward through the other cleaners and finally was removed into the tailing water from the rougher. The maximum effect possible was thus obtained from the dichromate.

Equipment for carrying out this system in a mill would be rather elaborate, as each middling would have to be handled individually and returned to its proper place in the circuit. Furthermore, soluble salts might accumulate in the circuit and have a detrimental effect on the flotation, as several soluble salts are formed while the dichromate is being broken down, and the dichromate is passing out relatively slowly with the tailing water. Not only was the maximum effect of the reagent obtained, but no clotting occurred in cleaning, probably because of the acid dichromate present in all parts of the circuit.

Several tests using the countercurrent system of cleaning were made in the earlier part of the work on low-grade table concentrates, which were quite free from oil rock. Results from these runs may be summarized briefly as follows: On low-grade table concentrates assaying about 20 per cent zinc a concentrate assaying 60 to 65.2 per cent Zn, 0.5 to 3 per cent Fe, a few hundredths per cent Pb, and less than 1 per cent CaO, was made; the tailings generally assayed 1 per cent or less Zn, with a recovery of 95 per cent. The obstacles encountered when oil rock was present in the flotation feed finally caused abandonment of the method. As discussed on pages 35 and 36, a water-cleaning treatment was necessary to remove the oil rock before the marcasite, not already removed, was acted on by the dichromate.

SECOND TAILING.

Roughing was the only treatment to discharge a finished tailing. As the blende floats freely after roughing it might be advisable to change the first cleaner into a second rougher, and to discharge therefrom a second tailing low in zinc. This method would relieve the rougher of part of its load and a high-grade concentrate would be made more easily. In all the tests, however, the tailing assayed 4 to 11 per cent Zn, thus preventing high recoveries, probably on account of the diluteness of the pulp (from 1:20 to 1:40) at the end of the second roughing. When the second tailing is thickened and refloated, a low-grade tailing is, of course, discharged. From information gained in these tests it was concluded that the making of a second tailing should be considered in treating an ore containing contaminants difficult to remove. In the tests reported no second tailing was made.

PROCEDURE FOR FLOTATION TESTS.

The procedure finally adopted for grading up the low-grade table concentrates and used in representative tests to be reported was as follows: The low-grade zinc concentrate that resulted from tabling sludge and fine tailings was stage ground. either with flint pebbles or iron balls, to pass 120 mesh. The amount of material—about 7,000 grams, or 15 pounds—that was taken for a test was divided into four charges. The ground material was wet when weighed, the first charge weighing about 2,500 grams and the other three each about 2,000 grams. The first charge was made heavier than the others, as no middlings were returned to it. In this way each charge weighed about the same at the beginning of the roughing. The hollow-shaft, mechanical agitation type of machine was used. The first charge was roughed in ordinary city water, in 4 to 1 pulp density, with No. 400 oil. Three to five drops of oil was added at intervals, about 45 to 50 drops being required for a roughing. As the weight of the drop of oil was known, it was easy to calculate the pounds of oil required per ton of feed. The tailing from the flotation rougher was a finished product unless an attempt was made to separate it into lime and marcasite.

The rougher overflow was returned for treatment, and about 1.5 c. c. of sulphuric acid was added to prevent clotting. The blende was then allowed to float, making an overflow and an underflow that was the first middling. The second cleaning was also a water cleaning, about 1.5 c. c. of acid being added to prevent clotting. No oil was added in the cleanings, as the charge had carried enough oil from the roughing to enable it to float freely. After two water cleanings, about 1 gram of sodium dichromate and 1 c. c. of acid was added to each of the third and fourth cleaners. One-half of this acid was added before the dichromate, and the other half was added in the dichromate solution. Underflows, or middlings, were made from each treatment, and the overflow from the fourth cleaner was taken to be the final zinc concentrate. These middlings were then dewatered, combined, and given a brief grinding in the pebble mill, after which they were sent to the roughing treatment of the second charge. In all tests performed the solids settled rapidly from the solution and left a clear supernatent liquid. The above operations were repeated in the roughing and the four cleanings, and resulted in the production of a second zinc concentrate.

The coarse blende dropped into the middlings and was ground to promote flotation in succeeding treatments. Grinding also broke up grains that for other reasons are not floatable. If the grinding were not performed, the coarser zinc either would build up in the circuit, creating a heavy circulating load, or it would pass into the tailings and thus increase losses. The iron in the middlings was also ground, but this risk was taken to insure high recoveries of zinc.

Table 13 gives screen analyses of middlings from a test where six cleanings were employed.

TABLE 13.—*Screen analyses of flotation middlings.*

Product.	Per cent of weight.	Assay, per cent zinc.	Assay, per cent iron.
First middling	100.0	6.2	20.80
+ 300-mesh	58.0	7.6	18.60
– 300-mesh	42.0	4.2	23.80
Second middling	100.0	7.9	18.50
+ 300-mesh	61.0	9.6	15.20
– 300-mesh	39.0	5.2	23.60
Third middling	100.0	16.9	14.85
+ 300-mesh	46.0	21.4	8.80
– 300-mesh	54.0	13.0	20.00
Fourth middling	100.0	32.0	10.00
+ 300-mesh	50.0	42.0	4.20
– 300-mesh	50.0	22.0	15.80
Fifth middling	100.0	32.0	11.40
+ 300-mesh	40.0	39.4	5.40
– 300-mesh	60.0	27.0	15.40
Sixth middling	100.0	48.3	5.80
+ 300-mesh	50.0	57.0	2.00
– 300-mesh	50.0	39.6	9.60

Table 13 shows primarily that the middlings advancing down the list carried increasing amounts of coarse zinc. Although these products were screened on 300-mesh screens, excessive grinding of middlings is not advocated. It is best to give a middling some treatment to change its characteristics before returning it to the circuit that produced it. It may be also seen in Table 13 that the coarser iron passed readily into the middling, whereas the fine iron was hard to remove from the zinc, as indicated in the iron analysis of the minus 300-mesh part of the sixth middling. Coarse iron was so well removed that practically only fine iron was dropped in this stage of the cleaning operations. The grading-up process in successive cleaners may be traced by the tabular analyses of the middlings. The table shows the necessity of having several successive cleaners in treating the Wisconsin products since marcasite is but gradually separated from sphalerite. This separation might be compared to fractional distillation, in which a product is redistilled or retreated several times in order to separate it into its pure constituents; each retreatment is a gradual step toward complete separation. Obviously, by less elaborate treatment the zinc concentrates produced by the present western milling processes could be equalled.

RESULTS OF FLOTATION TESTS ON LOW-GRADE TABLE CONCENTRATES.

The results of four representative locked tests follow. Many other tests were carried out, but these represent average results.

MILL NO. 1.

Results of the flotation of a table concentrate from sludge and fine tailings from the mill designated as No. 1 are contained in Table 14.

The assay of the feed or low-grade table concentrate shows that the zinc content was about what was sought in tabling. The iron assay is higher than that of the zinc, and as the insoluble content—much of which was oil rock—was high (5.2 per cent) this product constituted one of the most difficult encountered in the Wisconsin district. The flotation feed was composed of 77.8 per cent sulphides, the remainder being gangue. This percentage is higher than that of standard sulphides (60 to 75 per cent), but it may be accounted for largely by the high content of iron. In the upper part of Table 14 are the four concentrates and the four tailings resulting from the treatment of the four charges of ore in the test. The lower part of Table 14 is a résumé of the test showing average assays of the products.

TABLE 14.—*Mill No. 1. Flotation of table concentrate from sludge and tailings.*

Products.	Weight of products.	Per cent of total weight.	Lead.		Zinc.		Iron.		Lime.		Insoluble.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
	Grams.											
Feed	6,865	100.00	0.03	100.00	20.20	100.00	22.00	100.00	7.20	100.00	5.20	100.00
First concentrate	420	......	......	......	61.00	......	......	......	......	......	......	......
Second concentrate	370	......	......	......	61.20	......	......	......	......	......	......	......
Third concentrate	565	......	......	......	60.20	......	......	......	......	......	......	......
Fourth concentrate	635	......	......	......	60.60	......	......	......	......	......	......	......
First tailing	980	......	......	......	.60	......	......	......	......	......	......	......
Second tailing	1,080	......	......	......	.90	......	......	......	......	......	......	......
Third tailing	1,080	......	......	......	.80	......	......	......	......	......	......	......
Fourth tailing	1,070	......	......	......	.90	......	......	......	......	......	......	......
Zinc concentrate	1,990	29.00	Trace.	......	60.60	87.70	3.40	4.48	1.15	4.65	1.70	9.45
Middling	665	9.69	.10	28.31	20.60	9.86	18.00	7.93	8.50	11.49	9.50	17.64
Tailing	4,210	61.31	.04	71.69	.80	2.44	31.40	87.59	9.80	83.86	6.20	72.91
Total	6,865	100.00	......	100.00	......	100.00	......	100.00	......	......	......	100.00

Amount of reagents used:

Oil = 1.2 pounds per ton treated.
Acid = 11 pounds per ton treated.
$Na_2Cr_2O_7$ = 1.8 pounds per ton treated.
}**Cost equals 35 cents per ton treated.**

Zinc recovery $= \frac{C(H-T)}{H(C-T)} = \frac{60.6(20.2-0.8)}{20.2(60.6-0.8)} = 97.3$ **per cent.**

The average assay of the four concentrates from mill No. 1 should not, however, be considered as the grade of product possible to produce. Oftentimes the first, and possibly the second, concentrate assays higher in zinc than the succeeding products. The assay of the third or fourth concentrate is more likely to represent the correct grade, as middlings returned to the circuit have had an opportunity to become distributed. For this test the average grade of concentrates that could be produced with this amount of treatment is about 60.6 per cent Zn—the grade of the fourth concentrate. The same reasoning holds regarding the grade of the tailing.

The percentage of total weight of the middling is not correct as regards the amount circulating in a mill, which is usually between 15 and 25 per cent of the total feed. If this test had consisted of a single charge, the percentage of weight of middling then obtained would have approximated closely the percentage of material circulating in the circuit. On the other hand, if this test had comprised the treatment of a hundred charges of ore, the percentage of weight of middling then obtained would be fast approaching a negligible quantity. The assay of this middling, however, does represent the average grade to be expected from this feed.

The weight of middlings, or the ratio of middlings to the total feed, is independent of the number of cleanings. This statement applies to the present investigation, as clean zinc concentrates and tailings lean in zinc were made. If the grade of the rougher overflow and its ratio to the feed tonnage are known, the percentage of middlings can be ascertained, for the weight of the marcasite and gangue constituents of the rougher overflow equals the weight of the middlings. Thus, if the rougher overflow were 60 per cent in blende and amounted to 50 per cent of the feed by weight, the middlings would be 40 per cent of the 50 per cent—that is, 20 per cent of the feed if pure blende were made, or about 15 per cent if 90 per cent blende were made. The amount of middlings would probably never exceed $33\frac{1}{3}$ per cent, as the rougher overflow should not fall below 45 per cent blende and should not exceed in weight $66\frac{2}{3}$ per cent of the feed.

The recovery of zinc is not represented by the amount of zinc shown in the concentrate—87.7 per cent—as in this particular test the middling was holding up 9.86 per cent of all the zinc. Furthermore, it would not be correct to figure the recovery by absolute losses; this method would result in a recovery of 97.56 per cent (100—2.44). The middlings were holding some zinc that eventually passed into the tailings. The nearest approach to figuring the correct recovery is made by using the following formula:

$$\text{Recovery} = \frac{C(H-T)}{H(C-T)}$$

where C is the assay of the concentrates, H the assay of the heads or flotation feed, and T the assay of the tailing.

The recovery so calculated—97.3 per cent—is higher than the recovery already mentioned in this paper (p. 5). By sacrificing recovery, a higher grade of concentrate would doubtless be obtainable.

The iron content of the concentrate—3.4 per cent—was higher than in the other tests reported. An additional cleaning treatment would remove some of it; but as the four tests under discussion were performed in the same manner, this assay is here compared directly with the iron assays of the other concentrates.

The lead content in this flotation feed was too low to be given much consideration. The lime content of the concentrate—1.15 per cent—was well within the requirement for concentrates in the Wisconsin district. The insoluble content, due primarily to oil rock, was too high; but, as already pointed out, oil rock is exceptionally hard to remove, and the ore treated in this test is regarded as the most difficult in the district.

The flotation tailing was very high in iron—31.4 per cent. This tailing could be fed untreated to the sulphuric acid roaster, or it might be freed of some of the lime and insoluble constituents.

The cost of reagents per ton of feed may be determined by placing the cost of oil at 8 cents per pound, dichromate at 8 cents per pound, and acid at 1 cent per pound; the total being 35 cents per ton of flotation feed, or about three times that amount per ton of zinc concentrate produced.

MILL NO. 2.

Results obtained in floating a concentrate obtained by tabling sludge and fine tailings from another mill are contained in Table 15. The feed assays presented in Table 15 show that the zinc slightly exceeded the iron; the percentage of sulphides in the feed was 74.7. The assay of the tailings was consistent in the four charges; but the grade of the third concentrate dropped—probably because this concentrate was heavier than the others—indicating that in the last cleaning of the third charge too much material was allowed to float. A feature of ideal locked testing is that weights of all concentrates are equal. The assay of the fourth concentrate—61.8 per cent Zn, was about what would be expected from this feed with the given amount of treatment. The lead assay of the feed was low, but it is important to note that the lead was being concentrated in the middlings. In continuous running the lead content of the concentrate began to increase, only a trace being contained in the tailings. Hence the middlings would have to be tabled for the removal of lead. They would be tabled, ground, and returned to the head of the roughing machine.

TABLE 15.—*Mill No. 2. Flotation of table concentrate from sludge and fine tailings.*

Products.	Weight of products.	Per cent of total weight.	Lead.		Zinc.		Iron.		Lime.		Insoluble.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
	Grams.											
Feed	6,650	100.00	0.09	100.00	21.70	100.00	19.70	100.00	8.60	100.00	6.50	100.00
First concentrate	410	......	......	......	63.00	......	......	......	......	......	......	......
Second concentrate	350	......	......	......	62.60	......	......	......	......	......	......	......
Third concentrate	710	......	......	......	60.40	......	......	......	......	......	......	......
Fourth concentrate	480	......	......	......	61.80	......	......	......	......	......	......	......
First tailing	865	......	......	......	1.00	......	......	......	......	......	......	......
Second tailing	965	......	......	......	1.00	......	......	......	......	......	......	......
Third tailing	1,015	......	......	......	1.00	......	......	......	......	......	......	......
Fourth tailing	1,025	......	......	......	1.00	......	......	......	......	......	......	......
Zinc concentrate	1,950	29.32	.06	20.40	61.70	83.20	3.00	4.47	1.00	3.41	1.50	6.71
Middling	830	12.48	.55	79.60	21.60	14.13	18.20	11.54	7.00	10.23	7.90	15.11
Tailing	3,870	58.20	Trace.	......	1.00	2.67	28.40	83.99	12.70	86.36	8.80	78.18
Total	6,650	100.00	......	100.00	......	100.00	......	100.00	......	100.00	......	100.00

Amount of reagents used:

Oil=1.3 pounds per ton treated.
Acid=13 pounds per ton treated.
$Na_2Cr_2O_7$=1.8 pounds per ton treated.

Cost equals 37.8 cents per ton treated.

$$\text{Zinc recovery}=\frac{C(H-T)}{H(C-T)}=\frac{61.7(21.7-1.0)}{21.7(61.7-1.0)}=97 \text{ per cent.}$$

The iron assay of the concentrate was slightly lower than in Table 14, but was still somewhat higher than is expected in good commercial work. The lime content of the concentrate, 1 per cent CaO, was low enough to meet all smelter requirements.

Comparison of the insoluble content of the various products in Tables 14 and 15 shows how difficult the insoluble matter, largely oil rock, was to remove, the middlings and tailings assaying about the same. The figures in Table 14 suggest that the insoluble might be segregating in the middling; but the insoluble content of the middlings was high merely because the other impurities, lime and iron, are more easily put into the tailing and in consequence the tailings assayed higher in lime and iron than the middlings.

Again, it should be emphasized that the insoluble matter, or oil rock, is the most difficult impurity to remove. It is doubtful whether the content of insoluble could be lowered with further cleaning; the lime and iron assays, however, could probably be cut down one-half. The tailings, it should be noted, increased in weight from No. 1 to No. 4. The third and fourth tailings were about the same weight, thus indicating that the middling had properly distributed itself by the time the third tailing was made. Because of its high iron content, this tailing without further treatment might be suitable for roasting for the manufacture of sulphuric acid.

The recovery of zinc was high. The reagents cost 37.8 cents per ton of material treated.

MILL NO 3.

Table 16 presents the results of the flotation of a table concentrate from the tailing products of a third mill. The feed did not assay 20 per cent zinc, as the tailings from which it was made were not classified before tabling. The iron content of the feed was somewhat lower than that of the feed in the first two tests; the lime was higher, and the insoluble was considerably lower.

The zinc concentrate was slightly richer than in the other tests. The lead assay was low, but was concentrating itself in the middlings, as in Table 15. Provision should be made in a commercial installation to table middlings for lead before they are returned to the flotation circuit. Much iron could be removed at the same time, thus lightening the burden on the flotation machine. In other words, it is desirable to have tabling of middlings in closed circuit with flotation. The lime and iron in this feed were removed readily, as they were intimately locked together. The insoluble assay of the concentrate was lower than in the other tests, but indicated the reluctance with which this impurity was removed.

TABLE 16.—*Mill No. 3. Flotation of table concentrate from sludge and tailings.*

Products.	Weight of products (grams).	Per cent of total weight.	Lead.		Zinc.		Iron.		Lime.		Insoluble.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	**6,595**	**100.00**	**0.34**	**100.00**	**16.30**	**100.00**	**17.50**	**100.00**	**13.10**	**100.00**	**2.90**	**100.00**
First concentrate	400	......	......	......	62.80	......	......	......	......	......	......	......
Second concentrate	310	......	......	......	63.20	......	......	......	......	......	......	......
Third concentrate	390	......	......	......	62.60	......	......	......	......	......	......	......
Fourth concentrate	400	......	......	......	62.20	......	......	......	......	......	......	......
First tailing	1,130	......	......	......	.60	......	......	......	......	......	......	......
Second tailing	1,130	......	......	......	.80	......	......	......	......	......	......	......
Third tailing	1,165	......	......	......	.80	......	......	......	......	......	......	......
Fourth tailing	1,100	......	......	......	.80	......	......	......	......	......	......	......
Zinc concentrate	1,500	22.75	.17	11.38	62.60	87.12	2.80	3.65	.90	1.56	1.00	7.97
Middling	570	8.63	2.53	64.35	18.40	9.73	20.00	9.90	10.15	6.70	4.20	12.72
Tailing	4,525	68.62	.12	24.27	.75	3.15	22.00	86.45	17.50	91.74	3.30	79.31
Total	6,595	100.00	......	100.00	......	100.00	......	100.00	......	100.00	......	100.00

Amounts of reagents used:

Oils = 1¼ pounds per ton treated.
Acid = 11 pounds per ton treated.
$Na_2Cr_2O_7$ = 1.9 pounds per ton treated.
} Cost equals 36.2 cents per ton treated.

$$\text{Zinc recovery} = \frac{C(H-T)}{H(C-T)} = \frac{62.60(16.3-0.75)}{16.3\ (62.6-0.75)} = 96.5 \text{ per cent.}$$

The flotation tailing was probably too low in iron and too high in lime for sending directly to the roaster without further treatment. It was rather low in zinc, so that a high recovery—96.5 per cent—was

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obtained. This recovery might have been slightly higher if the grade of the flotation feed had been 20 per cent zinc or better. The cost of reagents per ton was 36.2 cents—about the same as in the other tests.

MIXTURE OF TABLE CONCENTRATES.

Table concentrates coming from different mills to a central plant for flotation treatment would no doubt be mixed, hence it was uncertain whether a mixture of a number of table concentrates could be floated without unforeseen troubles. Table 17 contains the results of floating a mixture, in about equal parts, of the three table concentrates treated in tables 14, 15, and 16.

The lead assay of the feed seems to be discrepant, but the lead and iron had not been tabled out as thoroughly from this feed as from the feed for the tests with the other tables. This fact accounts also for the iron assay being a little higher. The zinc concentrate was comparable in grade to that in the other tests; in some respects it was better. The middling showed the same segregation of lead, emphasizing the need of closed-circuit tabling. The tailing was about the same grade as the others made, and the recovery, 96.3 per cent, was comparable to the other recoveries. The cost of reagents per ton treated was 35.2 cents. In fact, all features of this test paralleled those of the tests on the individual table concentrates. The important fact brought out in Table 17 is that mixtures of table con-

TABLE 17.—*Flotation of a mixture of table concentrates from Mills 1, 2, and 3.*

Products.	Weight of products (grams)	Per cent of total weight.	Lead.		Zinc.		Iron.		Lime.		Insoluble.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	7,070	100.00	0.42	100.00	18.90	100.00	23.10	100.00	7.50	100.00	4.70	100.00
First concentrate	425	......	......	......	61.00	......	......	......	......	......	......	......
Second concentrate	360	......	......	......	62.20	......	......	......	......	......	......	......
Third concentrate	360	......	......	......	63.60	......	......	......	......	......	......	......
Fourth concentrate	480	......	......	......	62.40	......	......	......	......	......	......	......
First tailing	1,135	......	......	......	.80	......	......	......	......	......	......	......
Second tailing	1,040	......	......	......	1.00	......	......	......	......	......	......	......
Third tailing	1,190	......	......	......	1.00	......	......	......	......	......	......	......
Fourth tailing	1,060	......	......	......	1.00	......	......	......	......	......	......	......
Zinc concentrate	1,625	22.98	.05	2.75	62.25	75.65	2.60	2.58	1.00	3.06	1.10	5.38
Middling	1,020	14.43	2.04	70.33	27.60	21.05	17.20	10.74	6.00	11.53	5.70	17.48
Tailing	4,425	62.59	.18	26.92	1.00	3.30	32.00	86.68	10.25	85.41	5.80	77.14
Total	7,070	100.00	......	100.00	......	100.00	......	100.00	......	100.00	......	100.00

Amount of reagents used.
Oil=1.2 pounds per ton treated.
Acid=12 pounds per ton treated.
$Na_2Cr_2O_7$=1.7 pounds per ton treated.
} Cost equals 35.2 cents per ton treated.

$$\text{Zinc recovery}=\frac{C(H-T)}{H(C-T)}=\frac{62.25\ (18.9-1.0)}{18.9\ (62.25-1.0)}=96.3 \text{ per cent.}$$

centrates may be floated with as good results as when individual table concentrates are treated. None of these concentrates—each of which had come from a different part of the mining district—showed any peculiarities that would prevent it from floating when mixed. The proper treatment in a central plant would be to mix products almost free from oil rock with those containing a considerable amount of it, thus permitting the production of a high-grade concentrate from the mixture.

COMMENTS ON FLOTATION OF TABLE CONCENTRATES.

The four tests described above indicate the results that might be expected in a flotation plant. Of the three table concentrates treated, one came from the northern part of the Wisconsin district, one from the southern part, and the other from an intermediate point. They represent the various characteristics of the ores of the district, excepting those ores that are freer of impurities. From some of these so-called cleaner ores flotation concentrates have been obtained assaying as high as 65 per cent Zn, with impurities in practically negligible amount. For obvious reasons these tests have not been reported.

The foregoing tests show that a concentrate assaying more than 60 per cent zinc, about 3 per cent iron, 1 per cent lime, 1 to 1.5 per cent insoluble, and a small amount of lead may readily be produced with the amount of cleaning represented. If more cleaning were done, a higher-grade concentrate could be made, but at higher cost.

The grade of concentrate that should be produced depends, however, on various economic factors. This investigation shows that a commercially salable concentrate can be made at moderate expense. The recovery of zinc is above the 95 per cent, as mentioned in the early part of this paper. The cost of reagents is 35 to 38 cents per ton of material treated, or much less than $1.25 per ton of zinc concentrates produced. One oil has proved successful and others have shown promise. Moreover, if No. 400 oil were washed with dilute sulphuric acid before use, the cost of reagents would probably drop to about 30 cents per ton treated.

BY-PRODUCTS FROM FLOTATION.

No definite figures have been obtained on the amount of by-products from the flotation plant. Sometimes the flotation tailing might be rich enough in iron sulphate to sell to a sulphuric acid plant; in such event more than 90 per cent of the iron in the flotation feed would be available for roasting. Just how rich in iron the tailing must be to be sold to a sulphide-acid plant is undetermined, but is perhaps 30 per cent iron or about two-thirds marcasite. When the flotation tailing is not rich enough in iron, it can be graded up either by sulphide flotation or by tabling. Flotation tests on this tailing with sodium sulphide as a sulphidizer indicated that the iron can readily be concentrated into a product assaying 40 per cent iron. Table tests also resulted in a high-grade iron concentrate. Probably 70 per cent of the iron in the feed could be recovered by either method; but tabling would be cheaper and more convenient.

A lime concentrate suitable for agricultural purposes could hardly be made from the flotation tailing, as the product remaining was low in lime and carried also at least 5 to 10 per cent iron, whether tabling or flotation was used.

Lead recovered as a by-product would be from the tabling of flotation middlings in closed circuit with the flotation machine, the amount recovered depending on the quantity present in the flotation feed or table concentrate. Only a small return could be expected. The iron recovered by tabling the middlings would be high grade—about 40 per cent. It should be remembered that although sodium dichromate oxidizes the surface of the marcasite, the effect is so slight that it has not impaired the value of the mineral for the manufacture of sulphuric acid.

HANDLING OF FLOTATION CONCENTRATES.

In many western mills the equipment for handling flotation concentrates is elaborate, but such equipment is not contemplated for the concentrates of the Wisconsin district. The Tri-State practice, in which flotation concentrates are tabled primarily for dewatering, will probably be adopted. A little lead and iron may also be removed in this procedure. In Tri-State practice the overflow and tailing water carrying much fine zinc is sent back to the flotation machine, whereas it would be better practice to send it to a settling tank for the fine zinc to settle. From dichromate solution the zinc settles with unusual rapidity.

OTHER PRODUCTS THAT CAUSE MINERAL WASTE IN THE WISCONSIN DISTRICT.

Attention in the present investigation was devoted almost entirely to losses of mineral in milling. Mineral is also lost in the dust of the Cottrell precipitators at roasting and magnetic separating plants. This loss is small and is extremely difficult to recover by flotation, as the iron present is excessively fine ferric oxide—hematite—which coats the blende particles. Also, the presence of soluble salts in the dust would render its flotation difficult. No tests were made on this product.

The roaster tailing, however, carries considerable zinc (it assays 5 to 7 per cent) and amounts to about 8 per cent of the zinc in the roaster feed. It could not be separated effectively on the table, and in the flotation machine poor results were obtained. The following reasons may be given for these results: When the roaster product was put into water, strong acidity, approximately 100 pounds per ton of material, was observed. The cost of neutralizing this acid in order that flotation might succeed would be prohibitive. The roaster tailing also contained large quantities of soluble salts, mostly acid, that deterred flotation, and would have to be washed out. The iron was in the form of magnetite and even in water the grains seemed to stick together, holding up blende, and thus hindering flotation. Furthermore, any material that floated after persistent effort was no richer in zinc than the feed. Such obstacles made the treatment of roaster tailing extremely difficult, if not impossible.

A more logical method of seeking to eliminate the present losses of zinc in the roaster tailing would be not to make such a tailing. It was primarily for the purpose of reducing losses in roaster tailings that the following flotation of jig concentrates was tried:

RESULTS OF FLOTATION TESTS OF JIG CONCENTRATES.

The method of flotation procedure in five tests of jig concentrates was the same as in the tests of table concentrates in Tables 14 to 17 The jig concentrates were obtained from widely scattered points in the Wisconsin district and represented those concentrates that were the more difficult to treat. This work on jig concentrates did not constitute much research but simply the performance of a number of tests in accordance with information obtained in dealing with table concentrates.

The jig concentrates were coarse—about minus seven-sixteenths inch—and required crushing before treatment. Most of them contain lead enough to be given consideration. They might have been crushed through 1-mm. mesh, classified by a two-way classification as mentioned on page 18, and then tabled to recover the lead. If crushed to pass 40 mesh, the jig concentrates would require no classification before tabling; they might even be crushed immediately through 120 mesh and the lead recovered by tabling the flotation middlings in closed circuit. In this plan, dichromate must be used during the flotation treatment to put the lead into the middlings. Some lead that got into the flotation concentrates might be removed in tabling, and be returned to the circuit, so that eventually the lead would get into the middlings.

TABLE 18.—*Mill No. 1. Flotation of jig concentrate.*

Products.	Weight of products.	Per cent of total weight.	Lead.		Zinc.		Iron.		Lime.		Insoluble.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
	Grams.											
Feed	6,570	100.00	0.27	100.00	23.60	100.00	26.2	100.00	3.00	100.00	2.60	100.00
First concentrate	400				62.60							
Second concentrate	510				63.20							
Third concentrate	485				63.40							
Fourth concentrate	650				62.60							
First tailing	840				.70							
Second tailing	885				.80							
Third tailing	950				.80							
Fourth tailing	1,030				.80							
Zinc concentrate	2,045	31.13	.45	52.58	63.00	83.04	2.90	3.45	.80	8.46	1.00	12.08
Middling	820	12.48	.65	30.45	28.40	15.02	22.00	10.49	2.60	11.04	4.60	22.27
Tailing	3,705	56.39	.08	16.97	.80	1.94	40.00	86.06	4.20	80.50	3.00	65.65
Totals	6,570	100.00		100.00		100.00		100.00		100.00		100.00

Amounts of reagents consumed:
11 pounds = acid per ton treated.
1.1 pounds = oil per ton treated.
2.0 pounds = $Na_2Cr_2O_7$ per ton treated.
} Cost equals 35.8 cents per ton treated.

$$\text{Zinc recovery} = \frac{C(H-T)}{H(C-T)} = \frac{63.\ (23.6-0.8)}{23.6(63.-0.8)} = 97.7 \text{ per cent.}$$

MILL NO. 1.

Table 18 gives results of a flotation test of jig concentrates from the same mill from which the table concentrates in Table 14 (p. 45) were obtained. In both Tables 14 and 18 the iron content of the feed was extremely high—26.2 per cent—higher than the zinc. The lime content of the jig concentrates was low as compared to that of a table concentrate. The grade of the four succeeding concentrates was quite consistent, the average being 63 per cent. The amount of impurities was similar to that in the concentrates from the tailing. The lead assay was a little higher than might be expected, but would be remedied by the tabling of the middlings. As far as its grade was concerned, the flotation tailing was an ideal product for roasting. The oil rock in the "sulphur rock" of the jig concentrates was difficult to separate from the zinc. The zinc assay of the tailing was low, 0.8 per cent, and the consequent recovery extremely high—97.7 per cent. The cost of reagents per ton treated was the same as for floating table concentrates made from tailing.

The zinc content of the jig concentrate was no higher than that of the concentrates obtained by tabling the fine tailings. If it is profitable to ship jig concentrate to a roaster for further treatment, it should also be profitable to ship the table concentrates to a plant for final grading up, as the flotation treatment should not cost more than the roaster treatment.

TABLE 19.—*Mill No. 2. Flotation of jig concentrates.*

Products.	Weight of products.	Per cent of total weight.	Lead.		Zinc.		Iron.		Lime.		Insoluble.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
	Grams.											
Feed	6,545	100.00	0.34	100.00	32.00	100.00	16.10	100.00	6.80	100.00	3.00	100.00
First concentrate	695	......	.21	......	63.00	......	2.60	......	......	......	......	......
Second concentrate	700	......	.20	......	62.80	......	2.60	......	......	......	......	......
Third concentrate	670	......	.24	......	63.20	......	2.60	......	......	......	......	......
Fourth concentrate	1,025	......	.85	......	61.00	......	3.20	......	......	......	......	......
Tailing	2,780	......	.12	......	1.20	......	......	......	......	......	......	......
Zinc concentrate	3,090	47.21	.43	59.23	62.30	92.01	2.80	8.23	.85	5.93	.50	8.00
Middling	675	10.31	.85	25.78	19.80	6.39	21.00	13.49	8.20	12.49	5.20	17.99
Tailing	2,780	42.48	.12	14.99	1.20	1.60	29.60	78.28	13.00	81.58	5.20	74.01
Total	6,545	100.00	......	100.00	......	100.00	......	100.00	......	100.00	......	100.00

Amounts of reagents used:
Oil = 1¼ pounds per ton treated
Acid = 11 pounds per ton treated.
$Na_2Cr_2O_7$ = 2.5 pounds per ton treated.
Cost equals 41.0 cents per ton treated.

Zinc recovery $= \frac{C(H-T)}{H(C-T)} = \frac{62.3(32.00-1.20)}{32.0(62.3-1.20)} = 98.1$ per cent.

TABLE 20.—*Mill No. 3. Flotation of jig concentrates.*

Products.	Weight of products.	Per cent of total weight.	Lead.		Zinc.		Iron.		Lime.		Insoluble.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
	Grams.											
Feed	7,180	100.00	0.71	100.00	18.75	100.00	23.10	100.00	7.95	100.00	3.10	100.00
First concentrate	485	......	.10	......	64.00	......	1.80	......	......	......	......	......
Second concentrate	390	......	.40	......	63.60	......	1.80	......	......	......	......	......
Third concentrate	550	......	1.80	......	62.00	......	2.10	......	......	......	......	......
Fourth concentrate	485	......	1.90	......	60.80	......	2.40	......	......	......	......	......
First tailing	1,210	......	......	......	.70	......	......	......	......	......	......	......
Second tailing	1,100	......	......	......	1.00	......	......	......	......	......	......	......
Third tailing	1,120	......	......	......	.90	......	......	......	......	......	......	......
Fourth tailing	1,130	......	......	......	.90	......	......	......	......	......	......	......
Zinc concentrate	1,910	26.60	1.10	41.21	62.50	87.98	2.20	2.53	.65	2.18	.90	7.80
Middling	710	9.89	3.90	54.32	17.20	9.07	22.40	9.58	7.20	8.97	5.50	17.72
Tailing	4,560	63.51	.05	4.47	.87	2.95	32.00	87.89	11.10	88.85	3.60	74.48
Total	7,180	100.00	......	100.00	......	100.00	......	100.00	......	100.00	......	100.00

Amounts of reagents used:
Oil = 1⅓ pounds per ton jig concentrate treated.
Acid = 12 pounds per ton jig concentrate treated.
$Na_2Cr_2O_7$ = 1.7 pounds per ton jig concentrate treated.
Cost equals 36.3 cents per ton treated.

Zinc recovery $= \frac{C(H-T)}{H(C-T)} = \frac{62.50\ (18.75-0.87)}{18.75\ (62.50-0.87)} = 96.7$ per cent.

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MILL NO. 2.

Table 19 gives the results of the flotation of jig concentrates from mill No. 2, the source of the table concentrates in Table 15. The jig product was crushed to pass 40 mesh and tabled to remove part of the lead, then it was stage ground through 120 mesh to prepare it for flotation. The flotation procedure was the same as in preceding tests.

The grade of the jig material was higher than in the other tests, approaching the average grade of jig concentrates in the district. The grade of the first three flotation concentrates was consistent and high—about 63 per cent Zn. The amount of lead and zinc in each of them was the same. The fourth concentrate, however, dropped in grade and carried much more iron and lead, probably because flotation proceeded too long in the final cleaning: its weight is much greater than that of any of the other three.

The tailings of this test were combined and a composite was assayed. The zinc content—1.2 per cent—was slightly higher than in preceding tests. In roughing a jig concentrate as rich as this the pulp becomes very dilute at the end of the run, and doubtless should be thickened somewhat before roughing ends. The resulting zinc concentrates contained more than 62 per cent Zn. Tabling the middlings in closed circuit would probably free the concentrate of the lead and some of the iron present. The lime and insoluble assays were low. Probably the insoluble did not contain as much oil rock as in some of the other tests. The recovery was exceptionally high—98.1 per cent; the cost of reagents, 41 cents per ton treated, was but a little higher than in preceding tests; it was lower if based on per ton of zinc concentrate produced.

MILL NO. 3.

Table 20 presents the results of a test of a jig product from Mill No. 3, which also supplied the material for the test in Table 16. This jig concentrate was not tabled to remove lead, but was immediately ground for flotation. Because of the sulphur rock present the grade of jig concentrate was probably as low as any made in the district. Analyses for lead and iron were made to determine the reason for the falling grade of succeeding concentrates. The lead increased rapidly from 0.10 per cent to nearly 2 per cent. The iron increased somewhat, the lead probably consuming the dichromate and the iron not being effectively retarded. Here appeared again the need for tabling the middlings in closed circuit. It is interesting to note that a jig concentrate can carry as much as 8 per cent CaO. The cost of reagents was 36.3 cents per ton treated. The recovery was much the same as in the other tests discussed, but would doubtless have been a little higher if zinc content of the flotation feed had been higher.

MILL NO. 4.

Table 21 contains the results of a test of a jig product from a fourth mill. The material was crushed to pass 40 mesh and tabled to remove some of the lead. Then it was ground for flotation. The concentrates were usually high grade, but succeeding ones fell somewhat, probably because the lead, which was 1.5 per cent in the fourth product was only a trace in the first concentrate. The last tailing was lighter than the others, as the fourth charge—from lack of material—was lighter than the preceding charges. The tailing was higher in zinc than usual, probably on account of the diluteness of the pulp at the end of the run. Had the pulp been thickened before the roughing was completed, the grade of the tailing might have been lower. The cost of the reagents was 42.8 cents per ton treated, and the recovery was 97.1 per cent. The results in Table 21 are much the same as in preceding tables.

TABLE 21.—*Mill No. 4. Flotation of jig concentrate.*

Products.	Weight of products, grams.	Per cent of total weight.	Lead.		Zinc.		Iron.		Lime.		Insoluble.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	6,135	100.00	0.51	100.00	29.70	100.00	19.20	100.00	4.85	100.00	2.15	100.00
First concentrate	600	......	......	......	64.60	......	......	......	......	......	......	......
Second concentrate	650	......	......	......	64.00	......	......	......	......	......	......	......
Third concentrate	520	......	......	......	63.40	......	......	......	......	......	......	......
Fourth concentrate	640	......	1.50	......	62.20	......	......	......	......	......	......	......
First tailing	700	......	......	......	1.20	......	......	......	......	......	......	......
Second tailing	760	......	......	......	1.60	......	......	......	......	......	......	......
Third tailing	840	......	......	......	1.80	......	......	......	......	......	......	......
Fourth tailing	615	......	......	......	1.80	......	......	......	......	......	......	......
Zinc concentrate	2,410	39.28	.62	47.42	63.55	84.09	1.90	3.88	.50	4.06	.50	9.18
Middling	810	13.20	1.60	41.41	30.00	13.34	19.00	13.05	4.25	11.58	3.20	19.75
Tailing	2,915	47.52	.12	11.17	1.55	2.57	33.60	83.07	8.60	84.36	3.20	71.07
Total	6,135	100.00	......	100.00	......	100.00	......	100.00	......	100.00	......	100.00

Amounts of reagents used:
Oil=1.6 pounds per ton jig concentrates treated.
Acid=14 pounds per ton jig concentrates treated.
$Na_2Cr_2O_7$=2 pounds per ton jig concentrates treated.
} Cost equals 42.8 cents per ton treated.

$$\text{Zinc recovery} = \frac{C(H-T)}{H(C-T)} = \frac{63.55\,(29.7-1.55)}{29.7\,(63.55-1.55)} = 97.1 \text{ per cent.}$$

TABLE 22.—*Flotation of a mixture of jig concentrates from mills 1, 2, and 3.*

Products.	Weight of products, grams.	Per cent of total weight.	Lead.		Zinc.		Iron.		Lime.		Insoluble.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	7,190	100.00	0.40	100.00	25.00	100.00	21.10	100.00	5.75	100.00	3.05	100.00
First concentrate	525	....	.10	......	61.80	......	3.00	......	......	......	......	......
Second concentrate	500	....	.21	......	62.80	......	2.00	......	......	......	......	......
Third concentrate	800	....	.85	......	62.00	......	2.60	......	......	......	......	......
Fourth concentrate	645	....	.82	......	62.40	......	2.60	......	......	......	......	......
Tailing	3,960	....	.10	......	1.20	......	33.20	......	......	......	......	......
Zinc concentrate	2,470	34.35	.55	47.56	62.20	85.60	2.60	4.17	.60	3.60	1.00	11.25
Middling	760	10.57	1.46	38.64	27.80	11.76	18.40	9.21	3.75	11.78	4.80	16.61
Tailing	3,960	55.08	.10	13.80	1.20	2.64	33.20	86.62	8.80	84.62	4.00	72.14
Total	7,190	100.00	....	100.00	....	100.00	....	100.00	....	100.00	....	100.00

Amounts of reagents used:
Oil = 1.2 pounds per ton treated.
Acid = 10 pounds per ton treated.
$Na_2Cr_2O_7$ = 2.2 pounds per ton treated.
Cost equals 37.2 cents per ton treated.

$$\text{Zinc recovery} = \frac{C(H-T)}{H(C-T)} = \frac{62.2\ (25.0-1.2)}{25.0\ (62.2-1.2)} = 97.1 \text{ per cent.}$$

MIXTURES OF JIG CONCENTRATES.

The same question arose in regard to jig concentrates as in regard to table concentrates, namely, whether a mixture could be treated by flotation as successfully as an individual product. Table 22 contains the results of a test of a mixture of jig concentrates from the first three mills discussed in Tables 18, 19, and 20.

The results in Table 22 practically duplicate those presented in Tables 18, 19, and 20 for similar tests. Hence a mixture of jig concentrates can be treated as easily as the individual concentrates. The authors believe there would be no particular difficulty in the flotation of a mixture of jig concentrates with table concentrates, though this test was not made.

These tests, represented in Tables 18 to 22, show what may be expected metallurgically from the flotation of jig concentrates. The financial profits involved are not discussed in this publication. A recovery of about 97 per cent is effected as compared with the present recovery of 92 per cent at the roasting and magnetic separating plants. Moreover, higher jig recoveries might be made, as less attention would need to be given to keep the percentage of lime within certain limits. On the other hand disadvantages exist, such as the penalties smelters impose on flotation zinc.

LOCATION OF THE FLOTATION PLANT.

As a single mill does not produce enough material to operate a flotation plant with economy, a central plant should be established to serve several mills.

This plant should be near the center of the mining area so that the hauls from the mills would not be too long.

Obviously an abundant supply of water is requisite, but in the Wisconsin district it may be obtained almost anywhere. If fresh water is used in all cleanings, it is estimated that a maximum of about 600 gallons per minute would be required for a plant treating 100 tons in 10 hours.

The flotation plant should be situated within a few hundred yards of the roasting plants, so that iron could be transferred readily from the flotation plant to the roaster, and for other reasons.

One factor, however, is opposed to such proximity. Sulphur dioxide, which escapes from the roaster into the air and is also produced in the piles of roaster tailing, is detrimental to the flotation of blende, and it also consumes dichromate. In fact, one of the so-called differential processes for the flotation of lead and zinc is based upon the principle that sulphur dioxide renders blende immune to flotation. Although this factor may not be serious, its possible importance should be investigated before locating a flotation plant close to a roaster. A laboratory flotation machine could be operated at various distances from the roaster in order to determine the effect of sulphur dioxide in the air upon the flotation of sphalerite. If the effect is detrimental, a little more dichromate might neutralize the sulphur dioxide. More difficulty would probably be met with sulphur dioxide if flotation were carried out entirely in a water solution without chemical reagents.

ELIMINATION OF BARITE BY FLOTATION.

As barite occurs in some mines of the district, the question has often been asked by operators: "Can barite be eliminated from the zinc by flotation?" A jig concentrate containing 28 per cent zinc, 15 per cent iron, and 19 per cent barite was stage-crushed through 120-mesh and roughed in the usual manner. It was then passed through the cleaning operations in dichromate solution, and the iron and barite were dropped into the tailing. The ease with which barite dropped into the tailing is shown by the fact that the zinc concentrate assaying 62 per cent zinc carried only 0.3 per cent barite. The recovery of zinc was good. The iron in the flotation tailing probably could not be separated from the barite on tables, as the specific gravities are about the same. The iron could be recovered by sulphidizing, and the barite could be eliminated from the blende by flotation.

FLOW SHEET OF THE FLOTATION PLANT.

In the following discussion of a general flow sheet of a flotation plant no attempt is made to specify the number and type of machines and such details. As explained above, a central plant should be maintained for several mills. Furthermore, a flotation plant for the proposed scheme of grading up table concentrates or jig products must be rather elaborate. The amount of cleaning involved in this process would astonish western millmen, but the justification for it lies in the very high grade of the products and the extremely high recoveries made. Western mills do not make as clean zinc products as these Wisconsin concentrates and usually their recoveries are low. The expense of cleaning as proposed in this paper is not great, and the result is high-grade products easily salable and highly desirable for further treatment.

Table concentrates would be prepared for flotation by grinding in tube mills in closed circuit. A large proportion of the concentrates—probably one-half—is already minus 120 mesh, hence the fines should be separated before the material is fed to the tube mill. If grinding with oil were practiced, all material would pass through the tube mill. The cost of grinding should be small, as the charge is 60 to 75 per cent sulphides, and the remainder (the gangue) is soft. Moreover, half of the charge is already the desired size. When jig concentrates are to be tabled for lead, the material could be ground by rolls and sent to the classifier for the tables. After the lead is tabled the product would be sent to the tube mills.

The hollow-shaft, mechanical-agitation type of flotation machines is recommended because it has proved successful in the laboratory. A vigorous action is required, especially in the roughing, to obtain a clean tailing. In the cleaning a milder agitating machine could be used. After being ground the material is fed to a roughing machine, which should have a number of cells, because the sphalerite is slow to float and time is required to get a lean tailing. The tailing from the flotation machine would either go direct to the roaster, to tables for the removal of lime, or to another machine for floating the iron. The sphalerite rougher overflow would pass to the first of a series of cleaning machines, the overflow from each passing to the next machine, and a finished concentrate would be taken from the last cleaner. These cleaners would probably be two-cell machines, as the material floats rapidly in cleaning. All the flotation machines should be arranged so that the overflow from one would fall by gravity directly into the next. The underflows from all cleaners would be combined, dewatered somewhat, and tabled for the removal of the lead and of some iron. Then these middlings would be given a brief grinding by themselves and returned to the head of the rougher. Tabling the middlings in closed circuit with the flotation machines is a feature that should be emphasized, as dewatering for tabling removes most of the soluble salts, thus preventing the accumulation of impurities in the circuit. The back water from the table would be the source of water supply for the rougher.

The most desirable number of cleaners is uncertain, but the minimum would be two using water alone, followed by two using dichromate solution. If a very high-grade concentrate were to be produced, more cleaners might be needed. They are relatively simple and cheap machines that could be decreased or increased in number according to several factors. The zinc concentrate could be tabled primarily for dewatering, also possibly for removing a small amount of lead-iron-zinc product, which could be returned to the rougher flotation machine. This tabling would dewater the bulk of the concentrates. The tailing and overflow water carrying much fine zinc should not be returned to the flotation circuit, but sent to a settling tank and then, perhaps, filtered. This fine zinc should not be sent back through the flotation circuit and then tabled again in imitation of Tri-State methods, as it will probably be eventually lost.

PUTTING RECOMMENDATIONS INTO PRACTICE.

In order to put into practice the recommendations made in this paper the procedure would be about as follows: In one of the mills a table unit should be placed to determine what can be done in making the various products from current material of the mill. If the tabling proved successful, it could be duplicated in other mills. Samples of the zinc concentrate obtained from this first unit could be floated in a laboratory machine by the procedure outlined in this report. Data on the results obtained by this unit and by particular mills would thus be accumulated.

After the table work was proved, a small flotation plant having a capacity of about 25 tons in 10 hours should be erected. Even though the table concentrates yielded no returns until treated by flotation, the lead, the marcasite, and the limestone tailing would be available to defray expenses. A small flotation plant is suggested, as this procedure in flotation is new and should be mastered on a small scale before being attempted on a commercial scale. It is bad practice to go from laboratory experiments to commercial operation in a single step.

After the flotation plant has been proved, an electrical plant for extracting zinc from flotation concentrates should be considered. In present practice metallic zinc is extracted from zinc concentrates by three principal methods: Fuel smelting, electric smelting, and the hydrometallurgic process. Fuel smelting of concentrates is both expensive and awkward, but is used for most of the zinc produced in the world. It is decidedly primitive when compared with parallel methods for the other metals. Because of the status of this method, many investigators have endeavored to perfect other methods, among them the electrolysis of sulphate solutions. The most successful method of recovering zinc from its concentrates hydrometallurgically is sulphuric acid extraction and electrolysis as adopted by the Anaconda Copper Co. in Montana.

Electrolytic recovery of zinc has rendered available for profitable exploitation great quantities of complex ores. For such ore the purification of solutions before electrolysis is extremely important. The impurities that are present in small amount in Wisconsin products are not particularly objectionable. A decided advantage of the electrolytic method is that the flotation concentrates through their fineness can be easily leached, although smelters penalize them for fineness. There would be no penalty imposed by a hydrometallurgic plant. The best zinc mineral for treatment by such a plant is sphalerite—the purer, the better. If the iron is not in solid solution or is not in too intimately crystallized with the zinc, it causes no difficulty in this process. Wisconsin concentrates are nearly ideal in composition and fineness.

A complete analysis of a mixture of flotation concentrates from Wisconsin jig concentrates and table concentrates follows:

Analyses of a mixture of flotation concentrates from jig and table concentrates.

Substance.	*Per cent.*	*Substance.*	*Per cent.*
Zn	62.83	Bi	None.
Pb.	.76	As	Do.
Fe	2.40	Sb	Do.
Cd	.21	Co	Do.
CaO	.15	Ni	Do.
MgO	.22	Al	Do.
SiO_2	.34	Mn	Do.
S	32.61	V	Do.
Insol. (incl. SiO_2)	.49	Cl	Do.
Cu	None.		

A sizing test showed that 81.5 per cent of this concentrate was finer than 200 mesh. It contained none of the real objectionable impurities—such as arsenic, antimony, and cobalt. The lead content could be lessened by tabling the middlings in the flotation plant. The small percentage of cadmium present can be removed from the electrolyzing solution without trouble.

Because of its purity electrolytic zinc has many special uses and it commands a higher price than ordinary zinc.

For commercial success in electrolyzing zinc, cheap power is a requisite. Hydroelectric power projects under way in Wisconsin may provide cheap power for this district, or it could be obtained at the Keokuk dam, Iowa.

Because of the need of greater efficiency in the utilization of the Nation's mineral resources, the authors of this report anticipate the erection of thermoelectric or electrolytic zinc-extraction plants in the Mississippi Valley. With such plants the Wisconsin zinc district would be metallurgically self-contained.

CONCLUSIONS.

The present recovery of zinc in the Wisconsin district can be increased 15.6 per cent by using concentrating tables in the mills, and by erecting central flotation plants. The loss is in the fine tailings. The grade of zinc concentrate would be 60 to 65 per cent zinc, not more than 3 per cent iron, 1 per cent lime, 1.5 per cent insoluble, and a few tenths of a per cent lead. The table recovery at the mills should be 80 per cent or better, depending upon the quality of the fine tailings. A central flotation plant should recover 95 per cent or more of the zinc in the flotation feed. The minimum recovery of zinc through the two operations should be 76 per cent.

The lead recovered on tables would be an important source of revenue. The fine tailings, the flotation feed, and the jig concentrates, if they are floated, would carry recoverable lead.

From the fine tailings a high-grade pulverized limestone may be made and marketed as a by-product.

From the fine tailings and from the flotation treatment another by-product, high-grade marcasite might be recovered for sale to a sulphuric acid plant.

The coarse sand from screening the original tailing would be available for concrete and road building.

The cost of reagents for flotation should not exceed $1.25 per ton of zinc concentrate made.

Jig concentrates can be graded up by flotation with a loss of only 3 per cent of the zinc, whereas in the existing method of roasting and magnetic separation the loss is 8 per cent.

Royalty on the use of flotation would have to be ascertained.

Smelters would penalize the "flotation zinc" ore about $5 per ton, but this penalty would be partly offset by the premium obtained when the grade went above 60 per cent. Nevertheless the penalty is discouraging because of the economic loss to the producer, and the implied loss to the country.

Under present roasting and smelting practice, 18.5 per cent of the zinc in the jig concentrates is lost before the zinc is cast into pigs. Flotation combined with electrolytic or possibly electrothermic methods of zinc extraction should eliminate most of this loss, and the increased recovery might be regarded as offsetting the higher cost of treatment. With an electrolytic or an electrothermic plant, the Wisconsin district would be self-contained metallurgically.

The advantages of the favorable climatic conditions and geographic location are great. The grade of zinc concentrate that it is reasonable to expect is so far superior to grades elsewhere that former criteria do not apply.

After roasting and leaching, there would be almost no gangue residue; any unroasted material could be sent back to the roaster.

Fuel heated retorts, such as are used in zinc smelters, are not adapted to the recovery of zinc from flotation concentrates; a better metallurgical process, such as the electrolytic, is required. It is in connection with inevitable improvements in the production of metallic zinc that the work described in this paper becomes most significant. If it is assumed that the metallurgy of zinc has taken rank with that of other metals, the problem in Wisconsin, as discussed in this report, assumes vast importance.

PUBLICATIONS ON METALLURGY OF ZINC.

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BULLETIN 202. Electric brass furnace practice, by H. W. Gillett and E. L. Mack. 1922. 319 pp., 25 pls., 35 figs.

BULLETIN 205. Flotation tests of Idaho ores, by C. A .Wright, J. G. Parmalee, and J. T. Norton. 1921. 70 pp., 8 pls., 1 fig.

TECHNICAL PAPER 283. Low-grade and complex ores of Colorado, by W. H. Coghill and C. O. Anderson. 1922. (In press.)

PUBLICATIONS THAT MAY BE OBTAINED ONLY THROUGH THE SUPERINTENDENT OF DOCUMENTS.

BULLETIN 154. Mining and milling of lead and zinc ores in the Missouri-Kansas-Oklahoma zinc district, by C. A. Wright and N. A. Buehler. 1918. 134 pp., 17 pls., 13 figs. 30 cents.

BULLETIN 171. Melting brass in a rocking electric furnace, by H. W. Gillett and A. E. Rhoads. 1918. 131 pp., 4 pls., 1 fig. 25 cents.

TECHNICAL PAPER 83. The buying and selling of ores and metallurgical products, by C. H. Fulton. 1915. 43 pp. 5 cents.

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